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VIP

AN OUTLINE OF THE METALLURGY OF IRON AND STEEL

PREPARED FOR THE USE OF STUDENTS
AT THE
MASSACHUSETTS INSTITUTE OF TECHNOLOGY
BOSTON, MASS.

BASED UPON PROFESSOR R. H. RICHARDS'
"NOTES ON IRON"

BY
einrich scar
PROFESSOR H. O. HOFMAN

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CHAPTER I.

INTRODUCTION.

SECTION 1. **Introductory**: Metallurgy is the art of extracting metals from their ores and refining and fashioning them for use in metal industries.

The subject is divided into three parts:—

1. *General Metallurgy*, discussing in a general way the physical and chemical properties of metals and their compounds, the metallurgical processes, ores, fluxes, apparatus, and metallurgical products and, in detail, fuels and refractory materials.

2. *Non-ferrous Metallurgy*, discussing in detail the treatment of ores of lead, copper, silver, gold, platinum, quicksilver, zinc, cadmium; tin, arsenic, antimony, bismuth, nickel, and cobalt; aluminum, uranium, tungsten, titanium, and other rare metals.

3. *Metallurgy of Iron and Steel*, discussing in detail the manufacture of these products.

Literature on the Metallurgy of Iron and Steel:—

(*a*) Manuals:—

Beckert, Th., "Leitfaden zur Eisenhüttenkunde." Second edition, Berlin, 1898, 1900, 190 .

Bell, Sir I. L., "Principles of the Manufacture of Iron and Steel." London, 1884.

Campbell, H. H., "The Manufacture and Properties of Iron and Steel." New York, 1903.

Dürre, E. F., "Die Anlage und der Betrieb der Eisenhütten." Leipsic, 1882–1892.

Gages, L., "Traité de Metallurgie du Fer." Paris, 1898.

Greenwood, W. H., "Steel and Iron." London, 1884.

Jüptner von Jornstorff, "Compendium der Eisenhüttenkunde." Vienna, 1896.

Jüptner von Jornstorff, "Grundzüge der Siderologie." Leipsic, 1900, 1902, 190 . Volume I translated by C. Salter. London, 1897.

Ledebur, A., "Handbuch der Eisenhüttenkunde." Fourth edition, Leipsic, 1903.

Percy, J., "Iron and Steel." London, 1864.

Sexton, A. H., "An Outline of the Metallurgy of Iron." Manchester, 1902.

Wedding, H., "Grundriss der Eisenhüttenkunde." Berlin, 1890.

Wedding, H., "Ausführliches Handbuch der Eisenhüttenkunde." Brunswick, begun 1891, not yet finished.

(*b*) Periodicals:—

Many physical, chemical, and engineering periodicals deal with matters relating to iron and steel. The following are devoted mainly to the subject:—

"The Iron Age," 1855 +. Since 1888 the text has been kept separate from the advertisements.

"The Iron Trade Review," 1867 +

"Jahrbuch für das Eisenhüttenwesen," 1900 +
"Journal of the Iron and Steel Institute," 1869 +
Kerpely, "Berichte über die Fortschritte in der Eisenhütten-Technik," 1864 +
"Stahl und Eisen," 1881 +
"The Iron and Steel Metallurgist and Metallographist," 1904 +

Chemically pure iron has no technical value. All industrial iron is a combination of iron with small amounts of other elements, such as C, Si, P, Mn, S, etc., which determine its physical properties. The most important is C. According to the carbon content industrial iron is divided into the great classes: cast iron, steel, and wrought iron. In the subjoined old classification, hardening power, which begins with about 0.3 per cent. C, was the distinguishing feature between steel and wrought iron. Since the advent of the open hearth furnace, in which iron with as little carbon as has wrought iron is produced, the term steel has been expanded so as to include melted iron with < 0.3 per cent. C, which is called mild or soft steel. With this enlarged meaning for the term steel the simple old classification suits modern conditions, and may serve as a modern classification. A more detailed modern classification is given by Howe in his "Iron, Steel, and Other Alloys," Boston, 1903, page 166.

KIND OF IRON.	CARBON CONTENT, PER CENT.	FORM.	MALLEABILITY.	HARDENING POWER.
Cast iron.	> 2.3.	Always cast.	Non-malleable.	Hardening.
Steel.	0.3 — 2.3.	Usually cast (slagless cast steel), rarely welded (slag-bearing weld-steel).	Malleable.	Hardening.
Wrought iron.	< 0.3 (usually < 0.1).	Never cast, always aggregated from pasty particles, slag-bearing.	Malleable.	Non-hardening.

CLASSIFICATION OF INDUSTRIAL IRON.

At the Centennial Exposition of 1876, in Philadelphia, the International Committee of the American Institute of Mining Engineers adopted the following classification:—

NON-MALLEABLE.	MALLEABLE.		
C(Si, P.....) over 2.3 per cent., melts suddenly upon heating.	C(Si, P.....) under 2.3 per cent., softens upon heating before melting.		
CAST IRON. Roheiſen. Fonte.	*Iron* (under 0.3 per cent. C.). Cannot be hardened by sudden cooling.	*Steel* (0.3 per cent. C. and over). Can be hardened by sudden cooling.	
	INGOT IRON. Flußeiſen. Fer fondu.	INGOT STEEL. Flußstahl. Acier fondu.	Cast when molten into a malleable mass.
	WELD IRON. Schweißeiſen. Fer soudé.	WELD STEEL. Schweißstahl. Acier soudé.	Aggregated from pasty particles without subsequent fusion.

CLASSIFICATION OF INTERNATIONAL COMMITTEE OF 1876.

This classification, in common use in Germany, Austria, etc., has not been accepted by England, United States, and France, where the term steel is made to include ingot iron.

THE WORLD'S PRODUCTION OF PIG IRON AND STEEL.[1]

METRIC TONS.

Year.	AUSTRIA, HUNGARY.		BELGIUM.		CANADA.		FRANCE.		GERMANY.		ITALY.		RUSSIA.	
	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.
1900	1,311,949	1,145,654	1,161,180	655,199	87,612	23,954	2,714,298	1,565,164	7,549,655	6,645,869	23,990	115,887	2,895,636	1,830,260
1901	1,300,000	1,142,500	765,420	526,670	248,896	26,501	2,388,823	1,425,351	7,785,887	6,394,222	25,000	121,300	2,807,972	1,815,000
1902	1,335,000	1,143,900	1,102,910	776,875	324,670	184,950	2,427,427	1,635,300	8,402,660	7,780,682	24,500	119,500	2,566,000	1,730,250

Year.	SPAIN.		SWEDEN.		UNITED KINGDOM.		UNITED STATES.		ALL OTHER COUNTRIES.		TOTAL.	
	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.	Pig iron.	Steel.
1900	289,788	144,355	526,868	300,536	9,003,046	5,130,800	14,009,870	10,382,069	625,000	400,000	40,198,892	28,342,294
1901	294,118	122,954	528,375	269,897	7,977,459	5,096,301	16,132,408	13,689,173	635,000	405,000	40,889,358	31,034,869
1902	278,000	124,000	524,400	283,500	8,653,976	5,102,420	18,003,448	15,186,406	615,000	412,000	44,557,991	36,479,783

The United States produced in 1902 40 per cent. of world's pig iron and 42 per cent. of world's steel.

[1] "The Mineral Industry," 1902, p. 306.

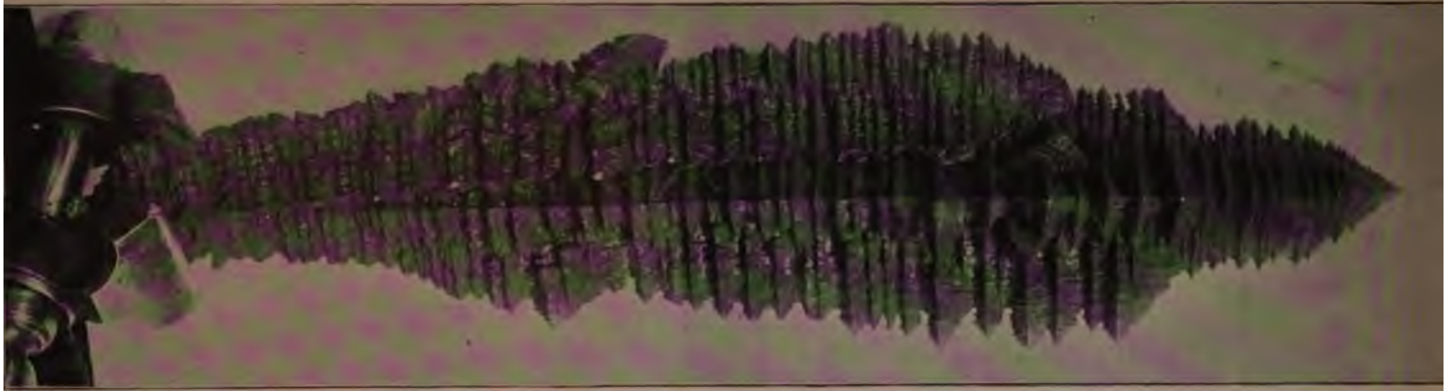
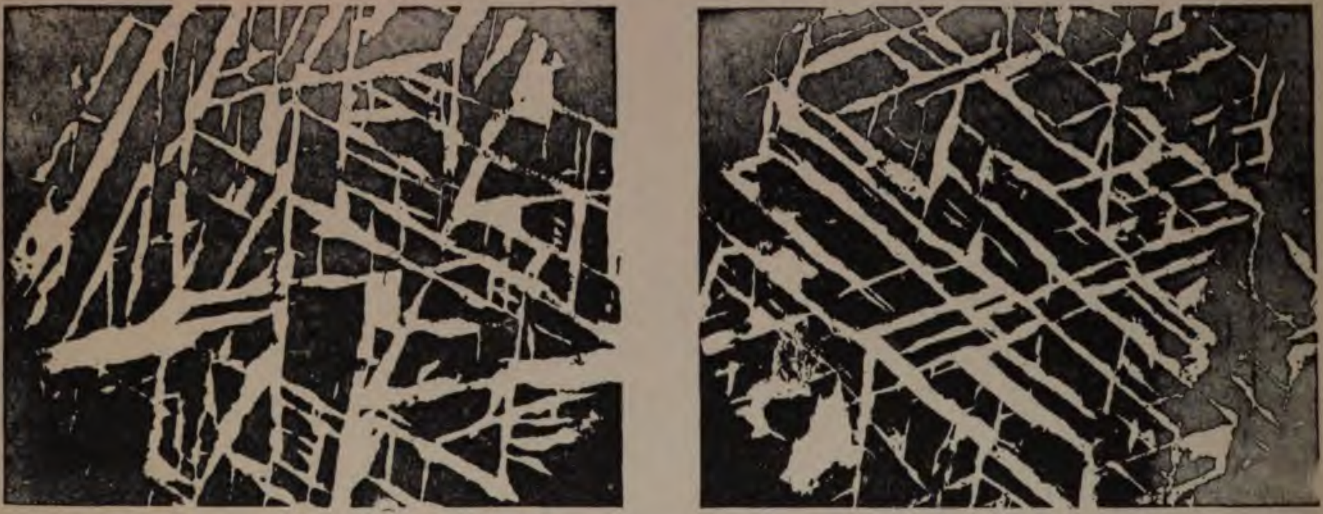


Fig. 2. Octahedral Crystal of Gamma Iron.



Figs. 3 and 4. Gamma Iron.

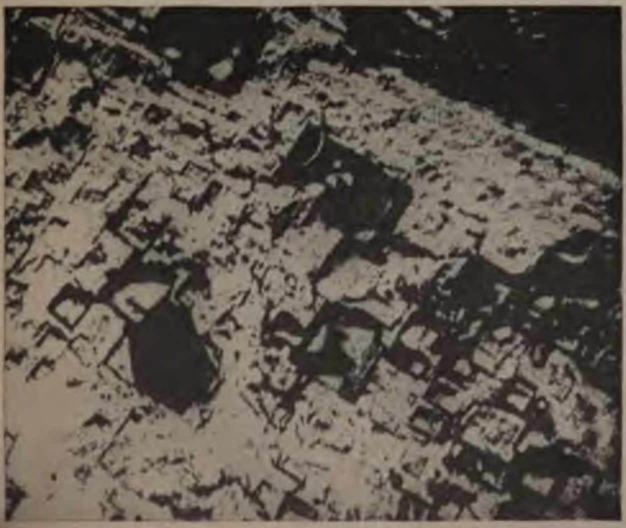


Fig. 6. Alpha Iron.

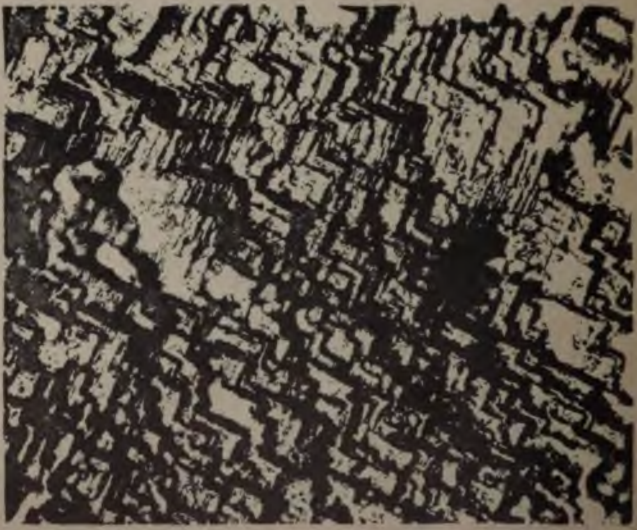


Fig. 5. Beta Iron.

SECTION 2. **Influence of Foreign Substances upon the Properties of Iron.**

Literature:—

General manuals quoted on page 1, further:

Howe, H. M., "The Metallurgy of Steel." New York, 1891.

Toldt, F., "Die Chemie des Eisens." Leoben, 1898.

Vosmaer, A., "The Mechanical and Other Properties of Iron and Steel." London, 1891.

1. *Chemically Pure Iron.* This is not used in the mechanical arts. It is gray, soft, crystallizes in hexahedral forms, and melts, according to Roberts-Austen,[1] at about 1,600° C. Others give 1,530° C. as the approximate melting-point.

If melted pure iron is cooled down slowly and the cooling-curve plotted, as shown in Figure 1, in which the ordinate represents degrees in the centigrade scale and the abscissa time in seconds, there will be observed jogs at 1,600° C., at 860° C., and at 750° C., which are due to evolutions of heat. The jog at 1,600° C. is due to the solidification of the iron; the retardations at 860° and 750° C. have been explained by Osmond by the iron assuming allotropic forms such as are known to exist in sulphur, phosphorus, etc. Between 1,600° C. and 860° C. (called Ar_3) iron has the form of *gamma*-iron (γ-iron), between 860° C. and 750° C. (called Ar_2) it has the form of *beta*-iron (β-iron), and below 750° C. it has the form of *alpha*-iron (α-iron). Ar_1, lying at 690° C., is the transformation point of carbonized iron (see later). The term Ar, given by Chernoff and Osmond, means *arrêt refroidissant* (an arrest in cooling), in contradistinction to Ac, *arrêt chauffant* (an arrest in heating). The two are not identical, Ac usually being about 30° C. higher than Ar (see later). In Roberts-Austen's cooling-curve[2] are given additional points of retardation and of recalescence (evolution of internal heat), denoting other molecular changes.

Γ-iron crystallizes in the isometric system (combinations of cube and octahedron); is non-magnetic; probably relatively hard, yet ductile; takes up at 1,000° C. about 1.5, not over 2 per cent. C; the dissolving power decreases with the temperature, so that at 700° C. it can take up only 0.8–0.9 per cent. C (see Figure 18). Characteristic for nickel steels and manganese steels with 12 per cent. Mn. Does not form an isomorphous mixture with β-iron. B-iron crystallizes in the isometric system in cubical forms; is non-magnetic; probably very hard and brittle; can take up very little carbon, less than 0.1 per cent. Probably characteristic for certain self-hardening steels, such as manganese steels with 7 per cent. Mn, and carbon steels when cooled suddenly from a red heat. A-iron crystallizes in the isometric system in forms similar to those of β-iron; is magnetic; is soft and ductile and relatively weak; can take up even less carbon than β-iron. Characteristic for wrought iron and low-carbon steels.

2. *Iron and Carbon.*[3] Iron can be carbonized by fixed carbon, by hydrocarbons, cyanogen, and cyanogen-compounds. Pure iron can take up 4.63 per cent. C. When supersaturated with C while liquid and cooled down slowly, flaky graphite (called kish) separates out and rises to the surface. The saturation point is raised by Mn (7.5 per cent. C) and Cr (7.8 per cent. C),

[1] Fifth Report, Alloys Research Committee, 1899, p. 48.

[2] Fifth Report, Alloys Research Committee, 1899, p. 35.

[3] Jüptner von Jornstorff, "Kohlenstoffformen im Eisen," reprint from "Ahrens Vorträge," V, p. 453. Stuttgart, 1896. Ledebur, "Journal of Iron and Steel Institute," 1893, II, p. 53.

and lowered by Si, S, and to some extent by P. The higher the percentage of carbon, the lower the melting-point of iron. Pure iron melts at 1,600° C.; iron with 4.1 per cent. C at 1,085° C. It was formerly believed that C (total carbon T. C.) existed in iron in two forms, as mechanically mixed graphite (G. C.) and chemically combined carbon (C. C.). It is now held that it exists in four forms, as graphite, temper carbon, cement carbon, and hardening carbon.

(*a*) *Graphite.* This is not present as such in molten iron, but segregates, uniformly distributed, in hexagonal plates with evolution of heat, when highly carburized iron solidifies. It is not attacked by acids, hot or cold, and is oxidized with difficulty by heating in air. Graphite forms mainly with highly carburized iron; is absent in iron that is malleable. Its formation is promoted by Si, Al, and slow cooling, and obstructed by Mn, S, and quick cooling (chilled castings). The influence of P is not certain. Hardness, tensile strength, and ductility are diminished by graphite; in the presence of much cement carbon and hardening carbon the tensile strength may be slightly increased by it.

(*b*) *Temper Carbon* (Temper Graphite). This separates out from carburized iron (> 0.9 per cent. C) running low in Mn, if the iron is held for some time at a temperature of 720–850° C.[1] in the form of a black amorphous powder. Its behavior with acids is the same as that of graphite. It is readily oxidized to CO by heating with Fe_2O_3. Mn retards its formation. The effects of Si, S, and P are not known. Its other behavior is the same as that of graphite. There is at present no method for separating temper carbon quantitatively from graphite.

(*c*) *Cement Carbon,* or Carbide Carbon (Fe_3C). This remains in the form of grayish-black, hard (Mohs scale 6), insoluble powdery grains with silvery luster when carburized iron is dissolved slowly in diluted H_2SO_4 with exclusion of air, or in dilute H_2SO_4 and K_2CrO_4. Boiling H_2SO_4 or HCl decomposes the carbide, C_xH_y being given off. Treated in the cold with HNO_3, it remains behind in the form of brown flakes, which dissolve upon warming, coloring the solution brown (Eggertz colorimetric method for C. C.). It is readily oxidized upon heating with oxidizing agents. It separates, with evolution of heat, from iron, whether highly carburized or not, at a temperature of 660–708° C. (Osmond), and disappears again when iron is heated above this temperature. Mn, W, as well as a high percentage of Si (favoring Gr. C.), and quick cooling (quenching) hinder its formation; small amounts of Si show no influence. Slow cooling (annealing) is favorable to its formation. It increases the hardness and tensile strength and reduces the ductility of iron. Campbell-Kennedy believe[2] that there exists a compound having the formula Fe_2C.

(*d*) *Hardening Carbon* is uniformly distributed in iron (alloyed with it). All carbon in iron when in the molten state is present as hardening carbon. It is decomposed by cold diluted H_2SO_4 or HCl, passing off as C_xH_y; with HNO_3 first a black flocculent residue remains, which upon warming and shaking passes into solution, coloring the solvent, and then passes off as gas, the solvent at the same time giving up its brownish color. The presence of Mn and quick cooling (quenching) favor its formation; the presence of Si and slow cooling (annealing) are unfavorable. Hardening carbon greatly increases the hardness and, up to a certain limit, the tensile strength of iron. It reduces the ductility.

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[1] Royalston, "Journal of Iron and Steel Institute," 1897, I, p. 166.

[2] "Journal of Iron and Steel Institute," 1902, II, p. 288.

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ASTOR, LENOX AND
TILDEN FOUNDATIONS.

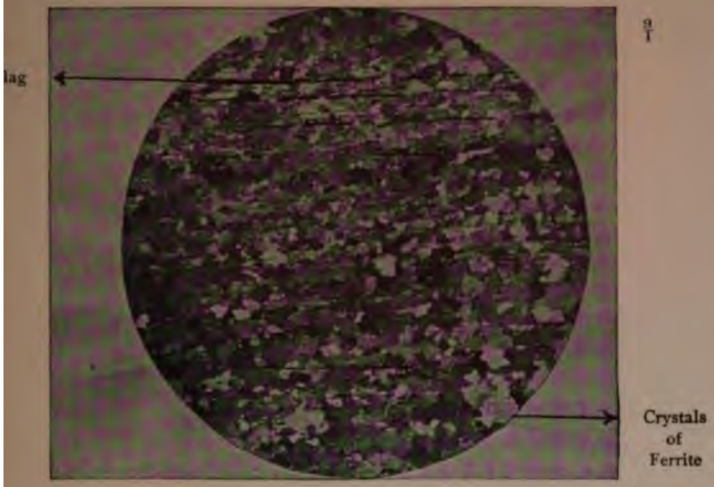

Fig. 7. Wrought Iron.

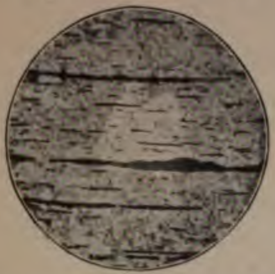

Fig. 163.
Wrought Iron.

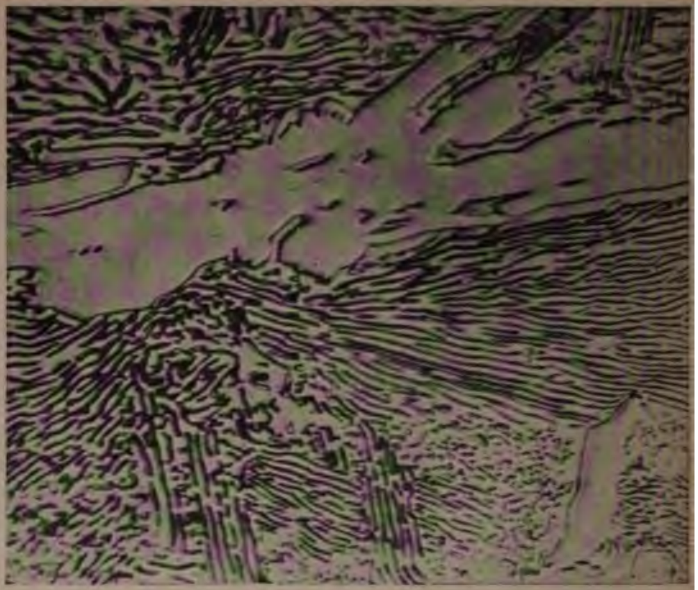

Fig. 10. Cementite and Pearlite.

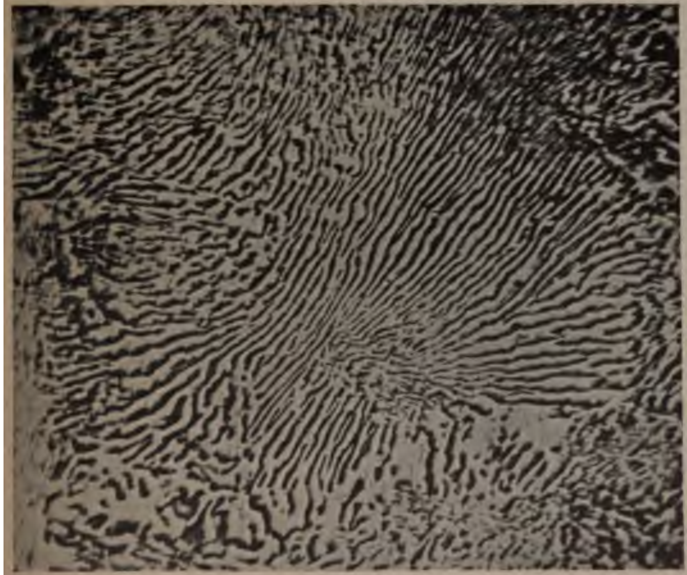

Fig 8. Pearlite.

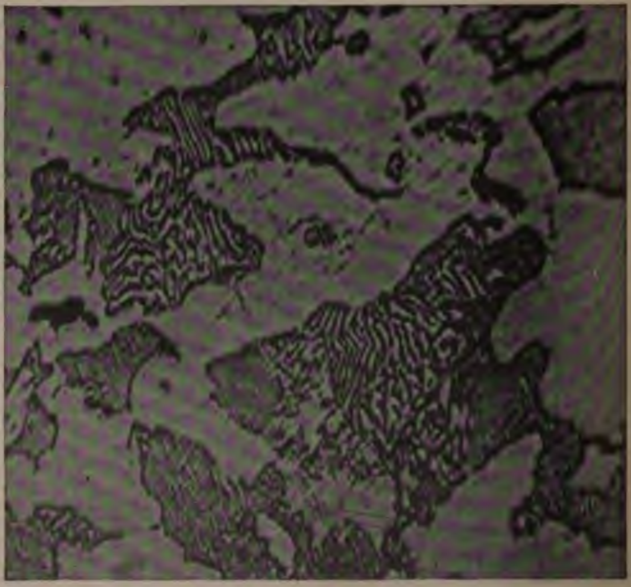

Fig. 9. Ferrite and Pearlite.

These different forms of carbon, being more or less regularly distributed through the iron, have varying effects on its structure. This is studied by the fracture and by a microscopical examination of polished sections. To the naked eye, or the eye fortified with a magnifying glass, the fracture will appear to be conchoidal, hackly, rough, smooth, or splintery. The microscopical examination[1] of metals was first suggested by Sorby[2] in 1864, who showed that the "various kinds of iron and steel are varying mixtures of well-defined substances," as in some igneous cryptocrystalline rocks the microscope reveals separate well-defined minerals not visible to the naked eye, *e.g.*, in felsite (devitrified obsidian) it shows clearly quartz and feldspar.

In commercial irons the following mineral constituents are distinguished: ferrite, cementite, pearlite, austenite, martensite, troostite, sorbite, graphite, and slag.

Ferrite[3] is either structurally free or forms a constituent of pearlite (see below). Figure 2[4] shows a crystal (15½ inches long) of γ-iron built up of octahedral forms resembling alum. The crystal, described by Chernoff, was formed in the pipe of a cylindrical ingot of soft, open-hearth steel weighing 100 tons, and cast with a sinking head 3.28 feet in diameter by 4.92 feet high. Figures 3 and 4[5] "show the structure of the sound region of the sinking head of a steel casting" with 0.45 per cent. C. The white ferrite lines cut each other at right angles, or more frequently form equilateral triangles which suggest the figures of Widmanstaetten,[6] in which crystallographers detect the characteristics of the regular octahedrons. The cubical form of β-iron is nicely shown in Figure 5, a photomicrograph by Stead[7] of iron with 4 per cent. Si. Figure 6, a photomicrograph by Heyn,[8] gives the cubical aspect of α-iron in soft steel with 0.25 per cent. C. Ordinarily (Figure 7)[9] the crystals of ferrite show under the microscope irregular forms, caused by interference with one another and preventing free development. The figure represents a longitudinal section through a piece of forged iron. The heavy black lines are elongated particles of slag; the rest of the field is taken up by ferrite.

Cementite is iron combined with cement carbon, answering the formula Fe_3C. It is intensely hard, occurs chiefly in refined white cast iron and spiegeleisen. It is found only with iron containing > 0.9 per cent. C, and is either structurally free or forms a constituent of pearlite. It exists thus in large masses, or in fine granules and thin plates.

Pearlite is the eutectoid formed at 690° C., when steel with 0.8–0.9 per cent. C is cooled

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[1] Howe, "Iron, Steel, and Other Alloys," Boston, 1903; Osmond-Stead, "Microscopical Analysis of Metals," London, 1904; Sexton, "An Outline of the Metallurgy of Iron," Manchester, 1902, pp. 520–560. Manipulations; Sauveur, "Engineering and Mining Journal," 1897, June 26, July 17, August 21, October 30; Stead, "The Metallographist," III, p. 220; Osmond, in "Contribution à l'étude des alliages, Commission d'alliages," 1896–1900, p. 277.

[2] "Metallographist," 1900, III, p. 89.

[3] Osmond, "Annales des Mines," 1900, XVII, p. 111; Osmond-Cartaud, *op. cit.*, 1900, XVIII, p. 113; abstracts in "Metallographist," III, pp. 181, 275; IV, pp. 119, 236.

[4] "Metallographist," 1899, II, p. 74.

[5] Osmond, "Metallographist," 1900, III, p. 201.

[6] Osmond-Cartaud, "Revue de Métallurgie," 1904, p. 69.

[7] "Journal of Iron and Steel Institute," 1898, I, p. 145; 1898, II, p. 137; or "Metallographist," I, p. 289; II, p. 85; III, p. 277.

[8] "Metallographist," 1900, III, p. 279.

[9] Sorby, "Journal of Iron and Steel Institute," 1887, I, p. 288.

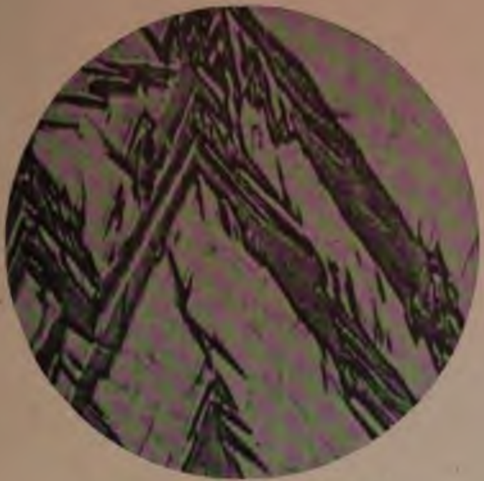

Fig. 11. Austenite and Martensite.

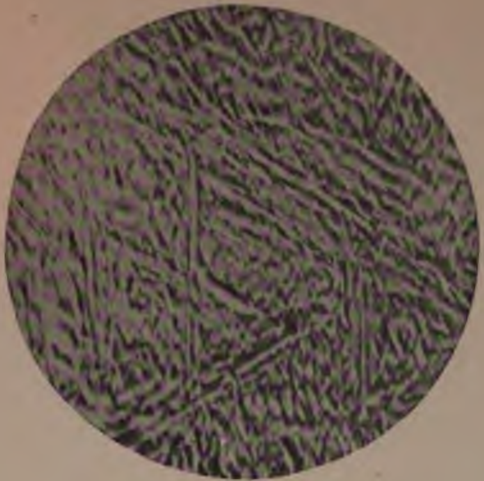

Fig. 12. Martensite.

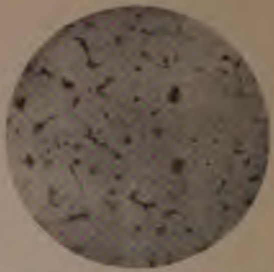

Fig. 20. Soft Bessemer Steel, 0.10 per cent. C.

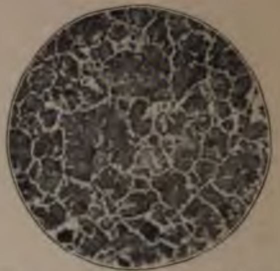

Fig. 21. Medium Hard Steel, 0.60 per cent. C.

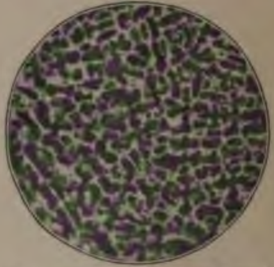

Fig. 22. High-carbon Steel, 2.50 per cent. C.

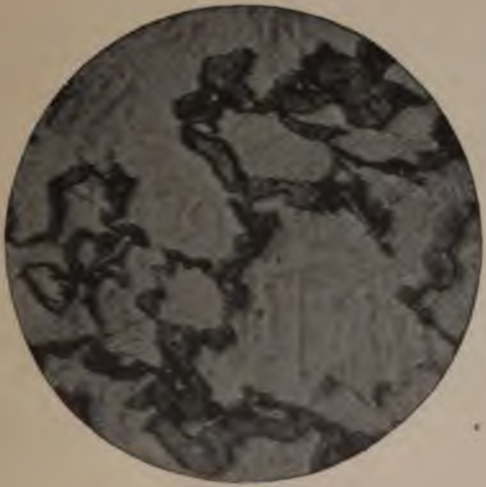

Fig. 13. Troostite.

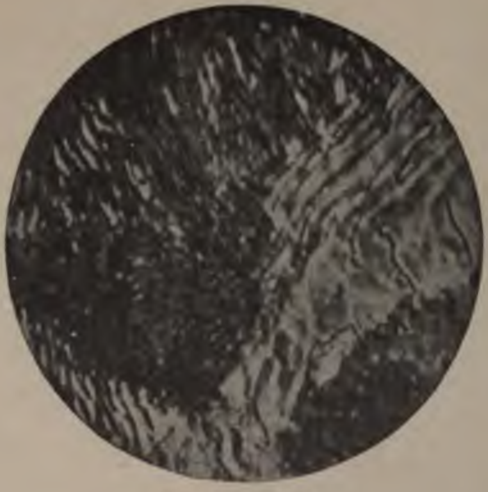

Fig. 14. Sorbite.

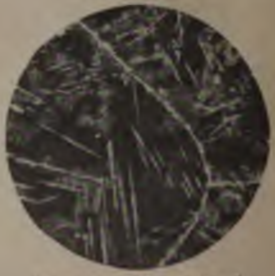

Fig. 313. Burnt Steel.

down slowly from a higher temperature. The term eutectoid[1] was given by Howe to the product of the lowest transformation in a solid alloy, in contradistinction to eutectic, which is the product that solidifies last upon cooling a fused alloy (see later).

Pearlite contains 88 per cent. ferrite and 12 per cent. cementite, assuming 0.8 per cent. C to be the saturation point of steel. Under a low magnification it appears to be a homogeneous substance; a high power reveals the two constituents arranged in alternating plates of cementite and ferrite, or in grains of cementite imbedded in pearlite. In Figure 8,[2] representing alternating plates, the dark lines are ferrite, the light lines cementite. This photomicrograph (1,000 diameters) by Osmond represents a steel with about 1 per cent. C which has been forged, reheated to 800° C., and then allowed to cool slowly. If a steel with less than 0.8–0.9 per cent. C is cooled down slowly below 690° C., the photomicrograph will show structurally free ferrite and pearlite, the ferrite areas increasing in proportion as the carbon decreases. In Figure 9 (forged steel, 0.45 per cent. C, 1,000 diameters, by Osmond[3]) the light areas are ferrite, the dark areas pearlite. If, on the other hand, a steel contains over 0.8–0.9 per cent. C, the photomicrograph will show cementite and pearlite. In Figure 10 (cement steel, 1.5 per cent. C, 1,000 diameters, by Osmond[4]) the light areas are cementite, the dark areas again pearlite. The reasons for the separation of the excess metal will be discussed later.

Austenite, Martensite, Troostite, and Sorbite (Figures 11–14).[5] *Austenite* is a solid solution[6] of carbon and γ-iron in varying proportions from 0 to 2 per cent. C. It is a hard, brittle, needle-like mass, which properties increase with the percentage of carbon. When iron with 2 per cent. or less carbon solidifies it forms austenite; upon cooling slowly it will be changed into martensite, troostite, sorbite, and this finally resolved into ferrite, cementite, and pearlite. If the carbonized iron is to retain its form of austenite it must be quenched before it has reached the martensite stage, and so on down the line. Austenite has a very strong tendency to change to martensite. Only by quenching a high-carbon steel very suddenly can it be retained to some extent in the cold metal. Thus Osmond obtained the structure shown in Figure 11 (1,000 diameters) by quenching in ice-water steel with 1.5 per cent. C heated to 1,110° C. The light ground mass is austenite; the dark zigzag constituent martensite. By quenching the same steel in ordinary water instead of in ice-water, martensite only would be obtained. With a low-carbon steel (say 0.45 per cent. C) even ice-water quenching will not stop the transformation halfway, but the whole will have changed into martensite. Figure 12 (1,000 diameters) represents a photomicrograph of steel with 0.45 per cent. C which has been heated to 825° C. and quenched at 725° C. It is composed wholly of *martensite*, having a needle-like triangular structure.[7]

In order to obtain *troostite* the cooling of the carbonized iron has to be retarded. This can be done by quenching in water (warm with small, cold with large, pieces) when the metal

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[1] "Engineering and Mining Journal," 1903: January 24, Howe; February 28, Brough; Marc [illegible]d: Marc Sauveur; March 28, Osmond; April 18, Howe, Grohman.

[2] "Metallographist," I, p. 1.

[3] "Metallographist," I, p. 21.

[4] "Metallographist," I, p. 1.

[5] "Metallographist," IV, p. 281.

[6] Sauveur, "Engineering and Mining Journal," March 21, 1903.

[7] A more recent illustration is given in the "Metallographist," IV, p. 240, Figur

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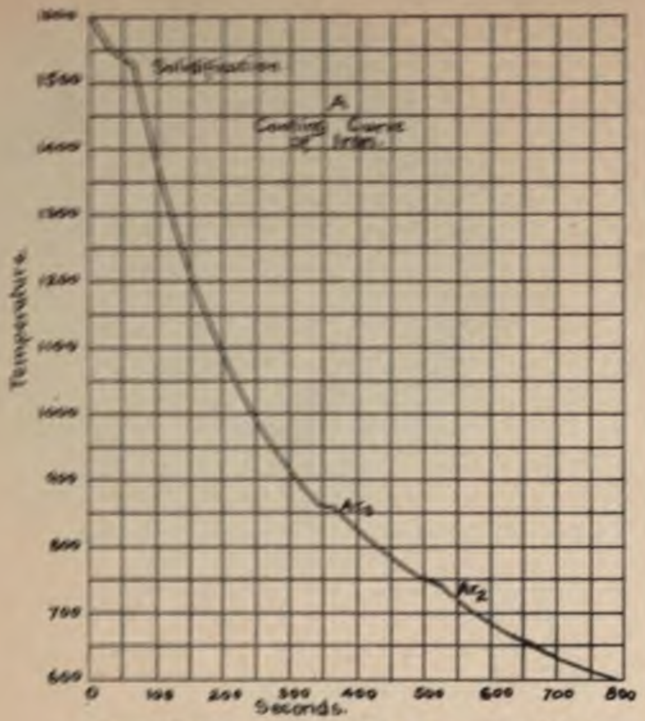

FIG. I COOLING CURVE OF PURE IRON.

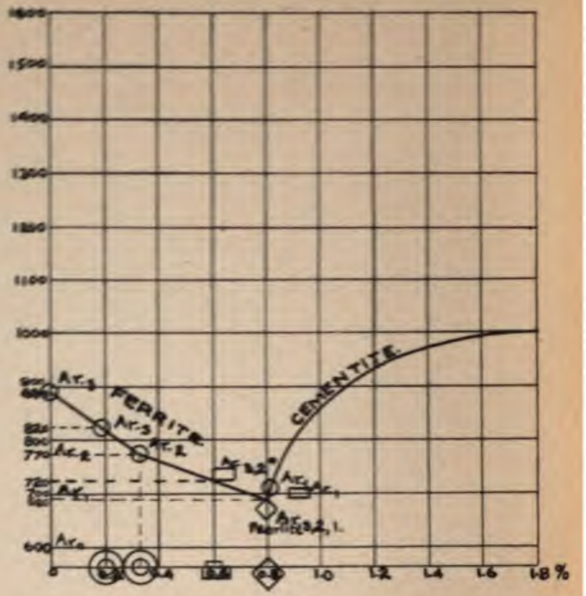

FIG. 17, DEPRESSION OF CRITICAL POINTS BY CARBON.

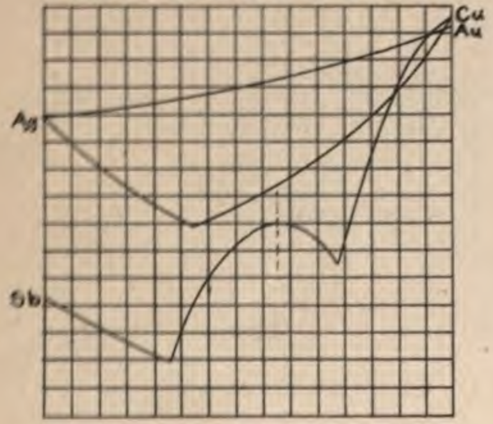

FIG. 15. TYPES OF BINARY ALLOYS.

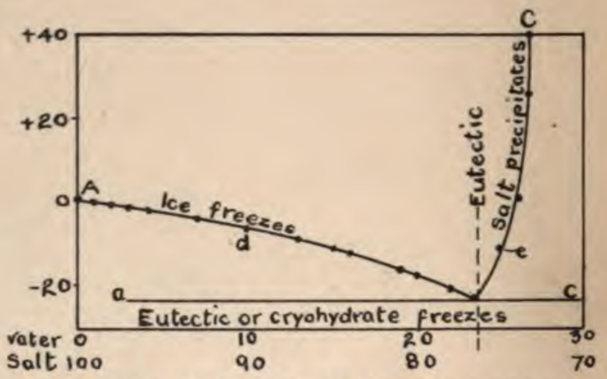

FIG. 16 FREEZING POINT CURVE OF SOLUTION OF SALT IN WATER.

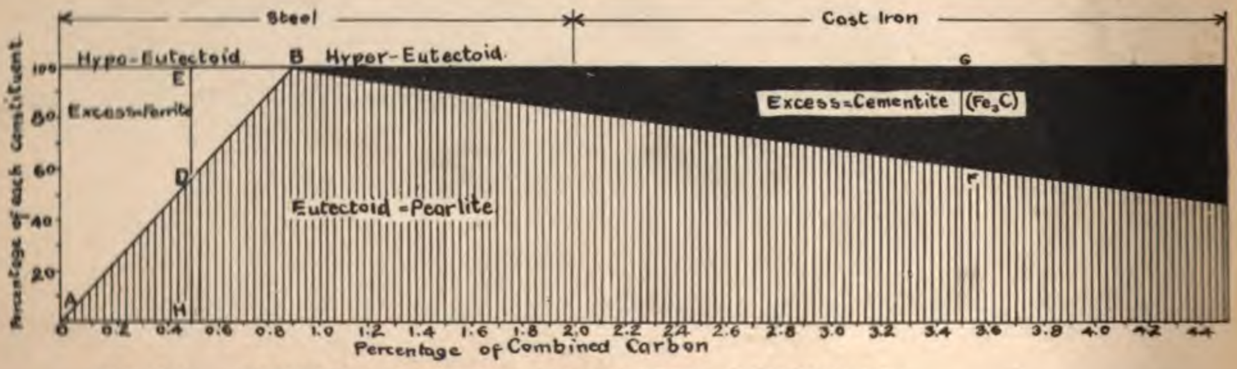

FIG. 23. CONSTITUTION OF IRON-CARBON COMPOUNDS.

passes through the critical range, or by quenching in oil from a temperature above the critical range, or by annealing martensite. In Figure 13 (1,000 diameters), steel with 0.45 per cent. C has been heated to 730° C. and quenched in water during the critical range. The dark, soft, vermicular parts are troostite; the matrix is martensite; here and there are small specks of ferrite. With a high-carbon steel (> 1 per cent. C) the ferrite would be replaced by cementite.

With *sorbite*,[1] a transition form between troostite and pearlite, the cooling must be still more retarded than is necessary with troostite. This can be done by quenching in melted lead at the end of the critical range, or by tempering the quenched metal to a purple color. In Figure 14 (2,000 diameters) steel with 1.24 per cent. C has been heated to 860° C. and cooled slowly in the furnace. The dark part is sorbite, the light part pearlite.

Howe,[2] in his "Iron, Steel, and Other Alloys," uses austenite as the generic name instead of martensite, and calls the latter a variety of the former.

The leading constituents of carbonized iron are ferrite, cementite, pearlite, and martensite; of minor importance are austenite, troostite, and sorbite, although sorbite is receiving more attention at present in connection with the rolling of rails.

Graphite is pure crystallized carbon, iron-black, with metallic luster; occurs chiefly in cast iron.

Slag is black; it is irregularly distributed in wrought iron and weld steel.

In Figure 18 are represented the leading constituents of the carbon-iron alloys and the changes they undergo when subjected to different temperatures. Before these can be understood, a few leading points about *Binary Alloys in General* have to be considered.

Literature: Roberts-Austen, "Introduction to the Study of Metallurgy," London, 1903. Books of Howe and Osmond (see *ante*). The "Metallographist," 1898, I, pp. 87, 192 (Charpy); I, pp. 137, 242, 342 (Roberts-Austen); 1902, V, p. 110 (Stead); V, p. 286 (Campbell). "Mineral Industry," X, p. 723 (Mathews). "Journal of Franklin Institute," January and February, 1902 (Mathews). "Reports Alloys Research Committee." "Transactions American Institute of Mining Engineers," XXXI, p. 528 (Fay-Gillson). "Mittheilungen d. Kgl. Versuchsanstaten zu Berlin," 1900, XVIII, p. 315 (Heyn). "Die Metallographie im Dienste der Hüttenkunde," by Heyn, Freiberg, 1903.

There are three classes of binary alloys (Figure 15):—

Class 1. The two constituents when solid are reciprocally soluble in all proportions. They form what is called solid solutions, and are consequently free from eutectics and do not form any chemical compounds. The freezing-point curve connecting the melting-points of the two metals forms a straight line, as shown by the *Ag-Au* curve. Other alloys of this class are those of Cd-Ag, Bi-Sb, Cu-Ni. A solid solution is a solid substance containing two or more elements in such intimate mixture that the highest power of the microscope cannot [illegible] them on properly prepared specimens.[3] The components of a solid solution can be as [illegible] [illegible]inguished from one another as when, *e.g.*, a liquid solution of salt in [illegible] is examined [illegible] microscope. Neither of the components can be seen separately. [illegible]s, see [illegible]

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[1] Sauveur-Boynton, "Transactions American Institute of Mining Engineers," [illegible] "Journal of Iron and Steel Institute," 1903, II, p. 141.

[2] "Engineering and Mining Journal," January 24, 1903.

[3] Stead, "Metallographist," 1902, V, p. 112.

Class 2. The two constituents are reciprocally insoluble or only slightly soluble in one another and do not form any chemical compounds. Such alloys form what is called eutectics. The V-shaped freezing-point curve of *Ag-Cu* is typical. The eutectic of an alloy is that which has the lowest melting-point. Its two other characteristics are (*a*) that its composition and hence its freezing-point are constant; (*b*) that it is not a chemical compound, but a distinct mechanical mixture of the two components. These are usually arranged in parallel plates not crossing one another (see Figure 8). Occasionally they have a spherulitic structure.

Modern investigations have shown that the structure of metallic alloys is governed by the same laws as the constitution of frozen saline solutions and of mixed melted salts. The following three laws of solution of solids in liquids are important for the study of the structure of alloys: —

(1) When a non-saturated solution of a solid in a liquid is cooled down to freezing the pure solvent alone separates out in the solid state.

(2) The lowering Δ of the freezing-point of a non-saturated solution of a solid in a liquid has been formulated by van't Hoff[1] $\Delta = \frac{n}{N} \times \frac{2T^2}{\lambda}$. It is proportional to the number of molecules of the dissolved substance, n, in a specified number, N, of molecules of the solvent, and to the quantity of $\frac{2T^2}{\lambda}$ which is dependent only on the nature of the solvent. (T is the absolute temperature of the freezing-point of the solvent, λ its latent heat of fusion.) With metals T is high (and T^2 especially so) as compared with aqueous solutions; the lowering of the freezing-point for a given concentration being greater, the higher the melting-point of the solvent. This shows how it is possible for mere traces of an impurity in a metal to have such a decided influence in lowering the freezing-point. One molecule of Ag dissolved in Zn lowers the freezing-point very much more than one molecule of Ag dissolved in Cu. The ratio would be $(415° \text{ C.} + 273° \text{ C})^2$ for zinc and $(1{,}054° \text{ C.} + 273° \text{ C.})^2$ for copper.

(3) When a saturated solution of a solid in a liquid is cooled down to freezing the composition of the solidifying portion is the same as the liquid remaining behind. The solid portion was called *cryohydrate* by Guthrie (1876), *κρύος*, meaning frost, ice cold.

Freezing-point Curves. Figure 16[2] represents the freezing-point curve for aqueous solutions of NaCl. It consists of two branches, *AB* (marked ice freezes) and *BC* (marked salt precipitates), meeting at a point corresponding to − 22° C. If, *e.g.*, a thermometer be placed in a 10 per cent. salt solution and this cooled down slowly by means of an external freezing medium, the temperature will sink uniformly until − 8° C. is reached, when there will be a sudden retardation, caused by the separation of crystals of pure ice free from salt. This gives the point *d* in the branch *AB*. The solution has now become more concentrated, and its freezing-point will be correspondingly lowered. The deposition of ice continues as the temperature is lowered until − 22° C. is reached, when the remaining liquid, containing now 23.5 per cent. NaCl, will solidify as a whole (cryohydrate), the temperature remaining stationary until the whole mass has solidified. The cryohydrate is not a chemical compound, but consists of crystals of ice and salt in juxtaposition. This is easily shown by using colored water, when

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[1] **Ostwald-Muir, "Solutions." London, Longmans, Green & Co., 1891, p. 222.**

[2] **Guthrie, "Philosophical Magazine," 1876, I, p. 359.**

the cryohydrate will be seen to be made up of colored plates of ice and colorless plates of salt or alcohol, which will dissolve out the ice, leaving a crystalline network of salt. The salt branch, *BC*, is steep, as the melting-point of salt lies at about 700° C. The point *e* represents a retardation at − 12° C., when from a solution containing about 25 per cent. NaCl crystals of pure NaCl will separate out. At − 22° C. the same cryohydrate as before will form. The cryohydrate always has the same composition, and solidifies at the same temperature.

The similarity between cryohydrates and alloys forming eutectics is clearly shown by *Ag-Cu* freezing-point curve (Heycock and Neville). In it *A* represents the melting-point of Ag, *C* that of Cu, and *B* the freezing-point of the eutectic alloy, corresponding to the cryohydrate of saline solutions. The name was given it by Guthrie (εὖ, well; τήκειν, to melt, melting readily or at a low temperature). If Ag containing some Cu (under 28 per cent.) be melted down in a crucible and cooled slowly, crystals of pure Ag will separate. The thermocouple of a Le Chatelier pyrometer plunged into it will show points of retardation. This will go on until 770° C. is reached, when the eutectic, with 72 per cent. Ag and 28 per cent. Cu, will solidify. The reverse will be the case if the start is made with Cu containing under 72 per cent. Ag, when Cu will separate out. A photomicrograph[1] of the eutectic will show crystals or plates of Ag and Cu in juxtaposition; one of an alloy of say 85 per cent. Ag and 15 per cent. Cu, when chilled, crystalline particles of Ag imbedded in a matrix of eutectic; one of say 35 per cent. Ag and 65 per cent. Cu, crystalline particles of Cu surrounded by eutectic.

The alloys of Au-Cu, Al-Zn, Sb-Pb, Zn-Bi, Zn-Cd, Zn-Sn, Cd-Sb, Sn-Pb, Sn-Bi, Fe-P, Fe-C belong to this class.

Class 3. The two constituents form one or more chemical compounds. The *Cu-Sb* freezing-point curve in Figure 15 forms a good example. There are three branches of curves, or $2n + 1$, where n represents the number of compounds. Two branches begin at the melting-points of the pure metals; the third crosses the former two at the eutectic points (75 per cent. Cu and 25 per cent. Cu) and shows a maximum in the summit. This represents the chemical compound $SbCu_2$ with 60 per cent. Cu. An alloy lying to the left of the eutectic (25 per cent. Cu + 75 per cent. Sb) will consist of meshes of Sb surrounded by a network of eutectic (25 per cent. Cu + 75 per cent. Sb); if it lies to the right, it will consist of islands of eutectic (25 per cent. Cu + 75 per cent. Sb) surrounded by a sea of chemical compound $SbCu_2$; if it lies at the summit of the roof-shaped curve, it will consist of the chemical $SbCu_2$ compound alone, and so on.

The alloys of Ni-Sn, Ag-Al, Zn-Sb, Pb-Cu, Pb-Al, Bi-Cu belong to this class.

Turning now to iron-carbon alloys. It has been shown that with pure electro-iron (Figures 1 and 17) the two points of retardation, Ar_3 and Ar_2, separating γ-, β-, and α-iron, are clearly marked. They become somewhat obscured when other elements combine with the iron. Take iron with 0.2 per cent. C (Figure 17). According to van't Hoff's formula, the freezing or retardation points ought to be lowered. Experiment has proved that Ar_3 has been lowered from 890° to 820° C. The second point, Ar_2, has been lowered, but only such a small amount as to practically remain in its original place. The third point, Ar_1, has remained unchanged. Take iron with 0.6 per cent. C. The point Ar_3 has been lowered to 720° C. The point Ar_2 ought to be lower down, but practically coincides with Ar_3, and is marked in the

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[1] See "Technology Quarterly," XI, p. 84.

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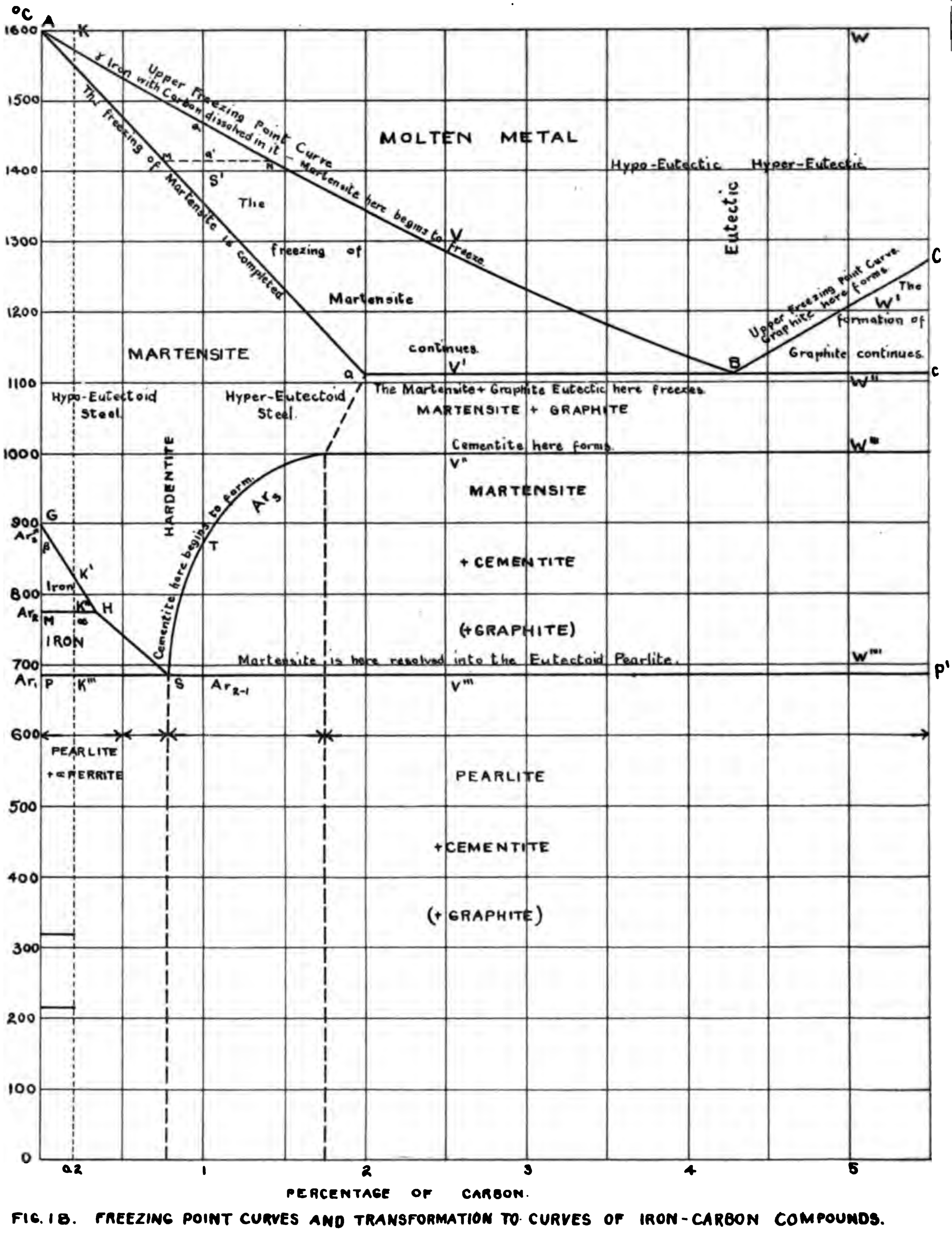

FIG. 1 B. FREEZING POINT CURVES AND TRANSFORMATION TO CURVES OF IRON-CARBON COMPOUNDS.

figure Ar_{3-2}. Ar_1 has remained unchanged. With iron containing 0.8 per cent. C, Ar_3, Ar_2, and Ar_1 coincide and are marked Ar_{3-2-1}.

If beside C the iron contains other elements, *e.g.*, Mn, the points of retardation Ar_3 and Ar_2 must be lowered still more, but they become also more obscured, so that with medium-carbon steels there are two retardations, Ar_{3-2} and Ar_1; and with high-carbon steels, one retardation, Ar_{3-2-1}. As far as commercial steels are concerned, the retardations, considered collectively, form a critical range, beginning at Ar_1. From what has been shown, it is seen that Ar_1 is the important point. The evolution of heat is sometimes so great as to produce an actual rise of temperature, a "recalescence" of the cooling metal, "temperature or point of recalescence." In passing through it the structure of steel is entirely changed, and with it all its other properties.

Other elements have an effect similar to that of carbon in shifting the critical points. This has been noted to be the case with Si, P, Mn, Cr, Ni, and Co. With nickel-steel of 25 per cent. Ni and manganese-steel of 13 per cent. Mn, Ar_3 is lowered 20° C.

In Figure 18[1] are recorded graphically the changes in constitution of iron-carbon alloys that may be caused by different temperatures. The ordinate gives the temperatures in degrees centigrade; the abscissa the percentages of combined carbon only, excluding graphitic carbon. The changes a cast iron with, *e.g.*, 3.5 per cent. total C (3.0 per cent. G. C. and 0.5 per cent. C. C.) undergoes will have to be looked for in the ordinate drawn on 0.5 per cent. C and not on 3.5 per cent. C. For the sake of convenience, iron-carbon alloys with 2 − per cent. C have been summarized as steel; with 2 + per cent. C as cast iron. The diagram is made up of two underscored V-shaped curves characteristic of eutectic and, as will be seen, of eutectoid alloys. The upper V-shaped curve *ABC* represents the points at which solidification occurs and the excess substance begins to separate out; the horizontal line *aBc* the region in which the eutectic solidifies. At *B*, representing 4.30 per cent. C, is the eutectic point. The line *Aa* represents the boundary at which solidification is completed. Thus above *ABC* there is only melted alloy; in the areas *AaB* and *CBc* there is a mixture of liquid and solid; below *AaBc* there is only solid alloy. The lower curve, *GHSER* and *PSP'*, represents the changes the solid metal undergoes upon further slow cooling. They are similar to those that take place in the upper curve, only they all take place in solid metal. A few new terms are used in the diagram: First, hardenite (Howe) is martensite with 0.9 per cent. C; if cooling is not arrested above Ar_1, hardenite will split into pearlite. Second, hypo- and hyper-eutectoid steel (Osmond-Howe); the former is steel containing less than 0.9 per cent. C, the latter more than 0.9 per cent. C. Third, hypo- and hyper-eutectic cast iron have similar meanings for 4.3 per cent. C. The meaning of the diagram is best made clear by following on their respective ordinates the changes that take place during slow and undisturbed cooling of a few carbon-iron alloys with different percentages of carbon.

Example 1. Hyper-eutectoid Steel with 1 per cent. C. At 1,600° C. it is molten. As the temperature falls, the line *AB* will be reached at *Q*, where a retardation is observed as the

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[1] **Roberts-Austen, Fifth Report, Alloys Research Committee. Roozeboom, "Zeitschrift für Physikalische Chemie," 1900, XXXIV, p. 437; "Journal of Iron and Steel Institute," 1900, II, p. 311; "Metallographist," III, p. 293. Stansfield, "Journal of Iron and Steel Institute," 1899, II, p. 169 ("Metallographist," III, p. 294); 1900, II, p. 317 ("Metallographist," III, p. 300). Howe, "Iron and Steel and Other Alloys," p. 192. Howe, Encyclopædia Britannica Supplement: "Iron and Steel" ("Metallographist," 1903, VI, p. 88).**

liquid solution of γ-iron and carbon begins to solidify and martensite begins to form. Between Q and S (on line Aa) the solidification continues, with the result of having a pasty mixture for liquid and solidified martensite. The cooling will proceed more and more slowly on account of the heat liberated by the solidification of martensite particles.

If we draw through Q', which lies in the partly liquid and partly solidified field, a horizontal line MN to meet the lines Aa and AB, the steel at the temperature Q' will consist of crystals of the composition M embedded in a liquid of the composition N, and the distances MQ' and $Q'N$ will represent the relative amounts of crystallized and liquid parts.[1]

At S there is a second retardation, as all the martensite is solidifying. Below S, until T (850° C.) on the line SE is reached, solid martensite cools uniformly. At T is the Ar_3 of the alloy, depressed from 890° C. by the 1 per cent. C. There is a third retardation, as part of the iron of the martensite begins to combine with the carbon to form Fe_3C (cementite). This goes on with the falling of the temperature until the carbon of the remaining martensite (the mother-metal) has been reduced to 0.9 per cent. C at about 690° C. (Ar_{2-1}) and has become hardenite; when passing below 690° C. this is resolved into pearlite. The end result is a mixture of cementite and pearlite.

Example 2. Hypo-eutectoid Steel with 0.2 per cent. C. The phenomena are at first similar to those of Example 1 from the time the alloy starts to cool at K until the ordinate reaches the line GH at K'. Here it crosses Ar_3 at 820° C., and the third retardation takes place. Some of the γ-iron of the solid solution crystallizes out and is at the same time changed into β-iron; the mother-metal becomes correspondingly enriched in carbon. As the cooling proceeds uniformly more iron separates out, until at K'' there is a fourth retardation, Ar_2 at 770° C.; β-iron changes into α-iron (α-ferrite). Iron continues to separate as α-iron until 690° C. has been reached. By the steady crystallizing of iron the carbon of the mother-metal has been increased to 0.9 per cent. C and hardenite has been formed. As soon as the temperature sinks below 690° C. the hardenite is changed into pearlite, and the end result is an intimate mixture of α-ferrite and pearlite.

Beyond 1.5 per cent. C the accurate position of the lines has not yet been definitely established, but approximately they are correct.

Example 3. Hypo-eutectic Cast Iron with 2.5 per cent. C. As solid γ-iron cannot hold in solution more than 2 per cent. C, the liquid alloy with 2.5 per cent. C will throw off some carbon in the form of graphite as soon as upon cooling it begins to solidify as martensite at the point V on the line AB. The remaining mother-metal becomes entangled with graphite. As the cooling progresses and more and more martensite separates out, the mother-metal becomes richer and richer in carbon, until at the point V' on aBc its carbon has reached 4.30 per cent., when upon further cooling it solidifies as eutectic of martensite + graphite, the heat liberated thereby causing a retardation. The solid alloy, consisting of martensite and eutectic (martensite + graphite), now cools further without interruption until 1,000° C. is reached on line ER. Part of the graphite combines with the iron of the martensite to form cementite, which change causes the retardation along the line. Below 1,000° C., between V'' and V''', the alloy consists of martensite and cementite and some residual graphite. In cooling from 1,000° C. to 690° C. cementite continues to form until the carbon of the martensite has been reduced to 0.9 per

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[1] See Heyn, "Die Metallographie im Dienste der Hüttenkunde." Freiberg, 1903, p. 8.

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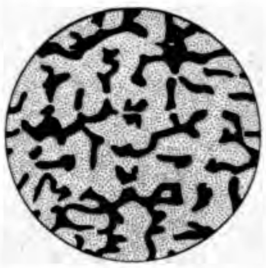
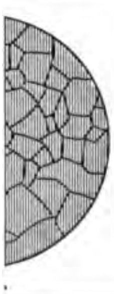


Fig. 15



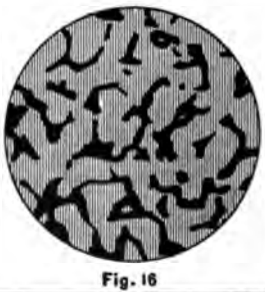


Fig. 16

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cent. (hardenite). This, upon further cooling, splits into pearlite. Thus below Ar_{2-1} the metal is composed of pearlite and cementite and some residual graphite.

Example 4. Hyper-eutectic Cast Iron with 5 per cent. C. Following the cooling along the ordinate W through $W'-W''''$, the changes are seen to be similar to those of a hypo-eutectic cast iron, with the exception that, containing more than 4.3 per cent. C, graphite will separate out at W' on the line BC upon cooling instead of martensite.

Some of the changes taking place in iron-carbon alloys with 0.09–2.50 per cent. C upon cooling are shown in Figure 19.[1] In order to retain the character of the different alloys they have been quenched.

Starting with 0.09 per cent. C, represented by subfigures 1–4, the steel when quenched above Ar_3 shows large (dotted) masses of martensite (77 per cent.) surrounded by a (white) network of ferrite (23 per cent.). When quenched between Ar_3 and Ar_2 the ferrite has had more time to separate out, and the polyhedric crystal forms of martensite have become smaller (27 per cent.) and the ferrite network larger (73 per cent.). Cooled between Ar_2 and Ar_1 the amount of ferrite (89 per cent.) has further increased. Cooled below Ar_1 there is a still larger network of ferrite (90 per cent.) and the martensite has changed to pearlite (10 per cent.). With steel containing 0.21 and 0.35 per cent. C (subfigures 5–10), retardations Ar_3 and Ar_2 have run together and the separation of ferrite begins only below Ar_2. Beginning with the eutectoid containing 0.8 per cent. C (subfigures 11–12), there is only one critical point. With steel containing more than 0.8 per cent. C cementite begins to separate out; the black particles in subfigures 13–14 are small; they increase with the percentage of C (subfigures 15–16).

The structural composition of steel can be easily calculated from its percentage of C. Let x = ferrite, y = pearlite, and z = cementite.

With steel containing	
Under 0.8 per cent. C.	Over 0.8 per cent. C.
$x + y = 100.$ $\frac{0.8}{100} y = \text{C}$ (per cent. carbon).	$y + z = 100.$ $\frac{0.8}{100} y + \frac{6.67}{100} z = \text{C}$ (per cent. carbon).

Figures 20–22 are low-power photomicrographs of slowly cooled steels. Figure 20, soft Bessemer steel, with 0.10 per cent. C, shows white ferrite with dark specks of pearlite; Figure 21, medium-hard steel, with 0.60 per cent. C, a light mesh-work of ferrite with a dark network of pearlite; Figure 22, high-carbon steel, with 2.5 per cent C, a light mesh-work of cementite with a dark network of pearlite.

Photomicrographs in high magnifications of ferrite, pearlite, cementite, austenite, martensite, troostite, and sorbite have already been given in Figures 8–14. Those of cast iron will follow later.

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[1] Sauveur, "Transactions American Institute of Mining Engineers," XXVI, p. 902.

A diagrammatical representation by Sauveur[1] of the constitution of slowly cooled iron-carbon alloys is given in Figure 23. The ordinate drawn on 0.9 per cent. C shows the steel to be composed wholly of pearlite; any ordinate drawn to the left of 0.9 per cent. C shows the percentage composition in ferrite and pearlite, and to the right in cementite and pearlite.

The effects of the total ferrite (the structurally free + that contained in pearlite) and of the total cementite (the structurally free + that contained in pearlite), as well as of the percentages of carbon in slowly cooled iron-carbon alloys, are shown in Figure 24.[2]

3. *Iron and Silicon.*[3] Silicon is readily reduced at an elevated temperature by carbon in the presence of iron (also of copper and silver), forming ferro-silicons which are dissolved by the iron. Iron with as much as 15 per cent. Si is produced in the blast furnace. The union of Fe with Si is accompanied by evolution of heat. According to Chalmot,[4] ferro-silicons produced in an electrical furnace with 25–50 per cent. Si appear to consist of mixtures of Fe_3Si_2 and $FeSi_2$. They are white and form isometric crystals not attacked by acids. Since then the researches of Guillet[5] have shown that iron and silicon form a solid solution and three chemical compounds, Fe_2Si, FeSi, and $FeSi_2$ SiO_2 is readily reduced by Mn without the intervention of carbon: $2Mn + SiO_2 = 2MnO + Si$. Si does not exist in iron in the graphitoidal state. It diminishes the power of iron to combine with C, and when present to an extent of at least 1.37 per cent. it forces part of the C to separate out as graphite. It slightly increases the fusibility and fluidity of iron, and to a greater extent the hardness; but less than C does. When present in insufficient quantity to affect the condition of the C, it increases the tensile and crushing strength (say up to 2 per cent. Si). Larger quantities make it brittle. It diminishes the ductility and forgableness, but very much less than C does. It reduces the conductivity for electricity and the magnetization, but is helpful for the retention of magnetism. It is readily removed from molten iron by means of oxygen, water vapor, iron oxides.

4. *Iron and Manganese.* MnO_X requires solid C and a white heat to be reduced to Mn, a carbide of manganese with 7.5 per cent. C being formed (Mn 90, C 7.5, Fe 2.5 .....) which is not attacked by acids. As Mn alloys in all ratios with Fe, the presence of reduced Fe favors the reduction of MnO_X. The smaller the percentage of Fe reduced with the Mn, the higher must be the temperature; and with ferro-manganese (90 per cent. Mn) the temperature becomes so high as to cause a partial volatilization of Mn, the fumes of which burn with a reddish-brown flame to Mn_3O_4. The 90-per cent. alloy soon crumbles giving off hydrogen. Mn has a greater affinity for C, S, and O than Fe. It therefore increases the power of C to combine with Fe, and thus counteracts its separation as graphite, which is favored by Si reduced at the high temperature required for Mn. Iron high in Mn is free from graphite. Mn is an effective agent for removing S and O from iron. On account of its affinity for O, manganese is readily removed from iron by an oxidizing smelting. A high percentage (20 per cent.) of Mn slightly raises the melting-point of Fe; a low percentage (1–2 per cent.) slightly lowers it. Iron

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[1] "Transactions American Institute of Mining Engineers," XXVI, p. 883.

[2] Howe, "Iron, Steel, and Other Alloys," p. 162.

[3] Hadfield, "Journal of Iron and Steel Institute," 1889, II, p. 222; Osmond, *op. cit.*, 1890, I, p. 62; Arnold, *op. cit.*, 1894, I, p. 107; Stead, *op. cit.*, 1898, I, p. 145; Baker, *op. cit.*, 1903, II, p. 312; Guillet, see above.

[4] "Journal of American Chemical Society," XXI, p. 59.

[5] "Revue de Métallurgie," 1904, p. 56.

with 20 per cent. Mn melts at 1,083° C.; with 50 per cent. Mn at 1,154° C.[1] Mn increases to a small extent the tensile strength and hardness (though not uniformly, viz., Hadfield steel) as well as the fluidity, raises the elastic limit and diminishes the ductility of iron. It reduces the electric ductivity and magnetization. Iron with a high percentage of Mn is non-magnetic. Guillet[2] finds that as regards the constitution of slowly cooled iron-manganese alloys three groups may be distinguished, viz. — First: C, 0.00–0.50 per cent.: Mn, 0–5 per cent., pearlite; Mn, 5–12 per cent., martensite; Mn, $>$ 12 per cent., solid solution. Second: C, 0.50–0.85 per cent.: Mn, 0–3 per cent., pearlite; Mn, 3–8 per cent., martensite + troostite; Mn, $>$ 8 per cent., solid solution. Third: C, $>$ 0.85 per cent.: Mn, 0–1 per cent., pearlite; Mn, 1–3 per cent., troostite and cementite; Mn, $>$ 3 per cent., solid solution.

5. *Iron and Phosphorus.* P_2O_5 is readily reduced by means of C at an elevated temperature in the presence of Fe, for which it has a strong affinity. Mn, Si, and even Fe will reduce P_2O_5 in the presence of a basic slag. In smelting iron ores in the blast furnace practically all the P enters the cast iron. P unites with iron in all proportions (at least up to 26 per cent.), but iron with over 3 per cent. P is rarely made. In dissolving phosphoric pig iron in diluted acid part of the P remains behind as Fe_3P (Mn_3P_2), part passes off as H_3P (as does the hardening carbon). Fe_3P is a crystalline, dark-gray, friable mass having a metallic luster. It is soluble with the evolution of H_3P in HNO_3, aqua regia and concentrated HCl. Stead,[3] in his study on iron and phosphorus, divides the alloy low in carbon into four classes: First, P, 0–1.7 per cent. P forms Fe_3P, which is held in solid solution by Fe. All wrought iron and low-carbon commercial iron belong to this class. Second, P, 1.70–10.2 per cent. This consists of a saturated solution of Fe_3P in Fe (*i. e.*, 1.70 per cent. P) and a eutectic (10.2 per cent. P) made up of 61 per cent. Fe_3P and 39 per cent. saturated solution of Fe_3P in Fe. (The name Steadite has been suggested for this eutectic by Sauveur.) Third, P, 10.2–15.58 per cent. This consists of crystals of Fe_3P surrounded by eutectic. Fourth, P, 15.58–21.68 per cent. Stead found a new chemical compound Fe_2P associated with Fe_3P, the Fe_2P increasing with the percentage of P. Carbon causes Fe_3P to separate out in part near the point of solidification when it comes in contact with phosphoric iron. The phosphide has the form of an eutectic in irregular-shaped areas when the percentage of C is low, of an envelope when high. Some phosphide always remains in solid solution, the amount decreasing as the carbon increases. Stead confirms (p. 123) the work of Arnold and MacWilliam,[4] that solid Fe_3P diffuses under certain conditions into solid iron, the amount being equivalent to 1 per cent. P. The temperature at which diffusion begins has not been determined. Arnold and MacWilliam, working at 1,000° C., found the diffusion of C and P to be as 5 : 1. P is readily removed from iron by an oxidizing smelting in the presence of basic slags; at a low temperature it is oxidized before the C, at a high temperature after it. P lowers the melting-point of iron, increases its fluidity, also slightly its hardness. Pure iron with 1.75 per cent. P has a hardness 5–6. P weakens the strength of iron. A low percentage of P has little effect on the tensile strength under a gently applied load; a high percentage reduces it greatly,

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[1] Osmond, "Comptes Rendus," CVI, p. 1,156.

[2] "Bulletin Société d'Encouragement," November, 1903 (abstract, "Stahl und Eisen," 1904, p. 281), and "Revue de Métallurgie," 1904, p. 91.

[3] "Journal of Iron and Steel Institute," 1900, II, pp. 60, 143 (literature).

[4] "Journal of Iron and Steel Institute," 1899, I, p. 85.

especially under shock-like strains. It increases the elastic limit, lowers the modulus of elasticity (?); makes the iron brittle, especially at ordinary temperature (cold-short). Cold-shortness is due probably to the coarsely crystalline structure of phosphoric iron. Carbon greatly intensifies the bad effect of P, Si to a less degree. P decreases the electric conductivity and the magnetization.

6. *Iron and Sulphur.* S readily combines with Fe up to 53.3 per cent. ($FeS =$ Fe 46.7 per cent., S 53.3 per cent., melting-point 950° C.[1]). The presence of CaO greatly counteracts the affinity of Fe for S. ($FeS + CaO + C = Fe + CaS + CO$.) In the absence of carbon the reverse is liable to take place. At a white heat, *e.g.*, iron will take away the sulphur from calcium sulphate: $CaSO_4 + Fe = FeS + CaO + FeO$.[2] MgO acts similarly, but much less effectively. The observation of Firmstone,[3] that a singulo-silicato-slag, in which CaO and MgO were present in the proportion of 1 : 2, acted as a better desulphurizer than when MgO was absent, may be attributed to the higher formation temperature of the magnesian slag.[4] MnO oxidizes S; Mn removes S as MnS; Si to a certain extent expels S. S causes cast iron to retain its C as combined carbon, but reduces the total C.[5] It lowers the melting-point of iron and makes molten iron thick and sluggish, and gives rise to blowholes. When not exceeding 0.1 per cent. it has no effect on the strength of iron when cold, making cast iron harder, and malleable iron tougher and softer; but when red-hot a much smaller amount than 0.1 per cent. causes red-shortness, and interferes with the welding. These effects are, to some extent, counteracted by Mn. At a white heat the effect of S disappears. It has been held[6] that iron sulphide readily diffused at 1,100° C. through solid iron. This permeability of iron is disproved by Le Chatelier.[7] Sulphur occurs in iron free from manganese as iron sulphide. Upon cooling iron from a high temperature the iron sulphide (melting-point 950° C.) solidifies last and separates out along the joints of the iron grains in the form of disconnected films if the percentage of S is low, of a connected network if high. It thus weakens iron, especially at a red heat, when FeS is in the fused state. If the iron contains manganese the sulphur will combine with it ($FeS + Mn = Fe + MnS$), but MnS is very little soluble in either iron or in manganese; the MnS, therefore, separates out in the form of isolated granules, resembling, perhaps, bubbles of gas, and, not breaking the contact of contiguous iron crystals, does not weaken the iron to any extent.[8]

7. *Iron and Copper.*[9] According to Stead[10] pure iron and copper can be alloyed by fusion in every proportion; carburized iron, however, takes up only small amounts of Cu, which segregate upon cooling in the form of microscopical pellets. As Cu is more easily reduced

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[1] Le Chatelier, "Metallographist," VI, p. 22.

[2] Hilgenstock, "Stahl und Eisen," 1893, p. 51.

[3] "Transactions American Institute of Mining Engineers," XXIV, p. 498.

[4] See also Forster, "Transactions American Institute of Mining Engineers," XXIX, p. 562.

[5] "Stahl und Eisen," 1903, p. 1,128.

[6] Campbell, "Journal of Iron and Steel Institute," 1897, II, p. 80; Arnold, *op. cit.*, 1899, I, p. 102.

[7] "Metallographist," VI, p. 29.

[8] See also Sexton, "Metallurgy of Iron and Steel," p. 553.

[9] Consult: Lipin, "Stahl und Eisen," 1900, pp. 536, 583, 691, 692; Colby, "Iron Age," November 30, 1899; Heyn, "Stahl und Eisen," 1902, p. 1,227.

[10] "Journal of Iron and Steel Institute," 1901, II, p. 104.

than Fe, and less easily oxidized, all the Cu in the smelting of iron ores enters the cast iron and cannot be removed by oxidizing smeltings, as is the case with Mn, Si, and C. Cu causes some red-shortness, as does S, only to a less degree, and destroys the welding power.

8. *Iron and Ni and Co, Cr, W.....* are of importance only in the manufacture of special steels, and will be touched upon then.

9. *Iron and Rarer Metals:* As, Sb, Bi, and Sn; Al; Ti, Ta, Va; Pb, Ag, Zn, Au, Pt, Pt-metals, Pd, Mo; alkali and alkali-earth metals are of minor importance.

10. *Iron and Gases:* H, CO, N, and O. Their absorption and effects will be treated under the special processes in which they show their influences.

CHAPTER II.

PRODUCTION OF PIG IRON.

Literature: General works quoted on page 1, further:

Bell, Sir I. L., "Chemical Phenomena of Iron Smelting." London, 1872.
Billy, E. de, "Fabrication de la fonte." Paris, 1895.
Gruner, L., translation by Gordon, L. D. B., "Studies on Blast-furnace Phenomena." Philadelphia, 1874.
Schinz, C., translation by Maco, W. H., and Müller, M., "Researches on the Action of the Blast Furnace." London, 1870.
Turner, "Metallurgy of Iron." London, 1895.

SECTION 3. **Introductory.** Pig iron is the technical name for the crude cast iron as it comes from the blast furnace. The raw materials required for the production of pig iron are ore, flux, and fuel. They are charged at intervals at the top of the blast furnace, filling it completely. Air under pressure enters at the tuyères and burns the fuel; the heat generated melts the charge, which undergoes chemical changes in its descent through the furnace. The products are crude cast iron (Fe with over 2.3 per cent. C, Si . . . . .) and slag ($Al_2O_3.xSiO_2 + CaO.ySiO_2$); they are tapped periodically from different levels in the crucible, and the hot gases (N, CO, CO_2 . . . . .) ascending in the furnace give up their heat to the cold descending charge, and, after causing various chemical changes to take place, pass off at the throat.

SECTION 4. **Iron Ores.**[1] An ore is a natural aggregation of minerals from which a metal or metallic compounds can be extracted with profit on a large scale. An ore consists of the metallic minerals and the gangue (non-metallic minerals). Leaving out *meteoric iron*,[2] which does not occur in sufficient quantity in one place to be called an ore, all iron minerals are essentially oxides.

I. THE IRON MINERALS. (1) Iron minerals proper: magnetite, Fe_3O_4, 72.4 per cent. Fe; hematite, Fe_2O_3, 70.0 per cent. Fe; limonite, $2Fe_2O_3.3H_2O$, 59.92 per cent. Fe; goethite, $Fe_2O_3.H_2O$, 62.93 per cent. Fe; turgite, $2Fe_2O_3.H_2O$, 66.29 per cent. Fe; siderite, $FeCO_3$, 48.3 per cent. Fe.

(2) Minerals which, after the extraction of one of their elements, become ores of iron: pyrite, FeS_2, 46.7 per cent. Fe (used for making sulphuric acid); franklinite, $(FeMnZn)O(FeMn)_2O_3$, about 46 per cent. Fe (used for making zinc white and spiegeleisen).

(3) Metallurgical products rich in iron: puddle-mill-heating-cinder. $mFeO_xnSiO_2$.

(4) Minerals harmful in iron smelting: rutile, TiO_2; ilmenite, $FeTiO_3 + Fe_2O_3$; pyrite, FeS_2; pyrrhotite, Fe_7S_8; gypsum, $CaSO_4 + 2H_2O$; barite, $BaSO_4$; chalcopyrite, $CuFeS_2$;

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[1] **Tenth Census** of the United States, 1880, XV; "Mineral Resources of the United States," 1882 +; "The **Mineral Industry,"** 1892 +

[2] **Osmond-Cartaud,** "Revue de Métallurgie," 1904, p. 69.

arsenopyrite, FeSAs; vivianite, $Fe_3P_2O_8 + 8H_2O$; apatite, $3Ca_3P_2O_5 + CaCl_2$ (or CaF_2); any others containing S, P, As, Ti, etc.

(5) Minerals used in making special kinds of iron and steel: pyrolusite, MnO_2, 63.3 per cent. Mn; psilomelane, $(BaMn)O + Mn_2O_3 + Aq.$, and others; chromite, $FeCr_2O_4$; wolframite, $(FeMn)WO_4$.

II. THE GANGUE of iron ores is siliceous, argillaceous, or calcareous, according to the prevalence of quartz, eruptive rocks, or clays, shales, slates, or limestones and dolomites.

III. STATISTICS.

1. WORLD'S PRODUCTION OF IRON ORES FOR 1900.[1]

Country.	Tons.	Country.	Tons.
United States	27,553,161	Sweden	2,609,500
Germany and Luxemburg	18,964,294	Cuba	444,840
Great Britain	14,028,208	Belgium	247,890
Spain	8,675,749	Canada	108,929
France	5,447,694	Other countries	1,455,772
Russia	5,000,000(?)	Total	88,064,478
Austria-Hungary	3,528,441	Percentage of United States	31.29

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[1] "Mineral Resources of the United States," 1901, p. 112.

2. PRODUCTION OF IRON ORES IN THE UNITED STATES FOR 1901.[1]

State or territory.	Hematite.	Limonite.	Magnetite.	Siderite.	Total.	Average value per ton.
	Tons.	Tons.	Tons.	Tons.	Tons.	
Minnesota	11,109,537	....	....	....	11,109,537	$1.38
Michigan	9,386,707	33,370	233,990	....	9,654,067	2.25
Alabama	2,070,422	731,310	....	....	2,801,732	.92
Pennsylvania	42,381	226,699	771,351	....	1,040,684	1.50
Virginia and West Virginia	13,156	910,214	2,024	....	925,394	1.58
Tennessee	314,949	474,545	....	....	789,494	1.16
Wisconsin	725,496	13,372	....	....	738,868	2.12
New York	66,389	23,362	329,467	....	420,218	2.39
Colorado	6,978	397,059	....	....	404,037	3.18
New Jersey	....	....	401,989	....	401,989	2.28
Montana, Nevada, New Mexico, Texas, and Wyoming	157,059	5,220	72,235	....	234,514	1.57
Georgia, North and South Carolina	60,102	153,477	2,020	....	215,599	1.20
Kentucky and Iowa	41,623	4,876	....	....	46,499	1.05
Ohio	....	....	....	....	44,185	1.53
Connecticut and Massachusetts	....	25,214	....	....	25,214	2.91
Maryland	....	14,993	....	....	21,218	1.59
Missouri	11,226	3,004	....	....	14,230	2.37
Total	24,006,025	3,016,715	1,813,076	51,663	28,887,479	1.71
Percentages for thirteen years	76.85	13.17	9.16	0.82	100.00	....
Percentages for 1901	83.10	10.44	6.28	0.18	100.00	....

The average yield of iron in ores used in 1899 was 51.27 per cent.;[2] in 1900, 53.2 per cent.[3]

IV. IRON ORES. (1) *Magnetite Ores.* Although magnetite is the richest iron mineral, the ores are not very rich, being much mixed up with gangue, and have often to be concentrated before they can be used. They contain 40–60 per cent. Fe. Magnetite ores are generally compact and massive and are most difficult to reduce. The gangue is, as a rule, siliceous, being quartz, hornblende, mica, epidote, garnet, chlorite, apatite, feldspar, etc.

The leading producers are Scandinavia, Russia, and the United States. In the United States magnetites come principally from New York, Pennsylvania, New Jersey, Michigan, and North Carolina. They form only a small percentage of the country's product of iron ore.

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[1] "Mineral Resources of the United States," 1901, pp. 45, 46, and 63.

[2] Eleventh Census, 1890, "Mineral Industries," p. 10.

[3] Twelfth Census, 1900, X, Part IV, p. 32.

P. — Scandinavian magnetite ores are low in P. Of the magnetites mined in the United States in 1880, 40.81 per cent.[1] were within the Bessemer limit.

In the acid Bessemer and acid open-hearth processes all the phosphorus of the pig iron goes into the steel, and this may not contain over 0.1 per cent. P. In smelting iron ore in the blast furnace all the P of the ore goes into the pig iron; thus an iron ore furnishing pig iron with less than 0.1 per cent. P is within the Bessemer limit; one that does not is outside of it. Of two ores with 0.4 and 0.6 per cent. P and 50 per cent. Fe, the first giving pig iron with 0.08 per cent. P will be within the Bessemer limit; the second, with 0.12 per cent. P, will not. In the example pig iron is assumed to contain 100 per cent. Fe, while according to the definition given on page 2 it contains less than 97.7 per cent. Fe.

S. — New York magnetites run low, Pennsylvania high in S (1–2 per cent.), and have to be roasted before smelting.

Ti is not uncommon. Titaniferous magnetites are of no commercial value at present, although in an experimental way they have been successfully smelted.[2]

ANALYSES OF MAGNETITES.

	ELEMENT.												
	Fe	P	S	Mn	P_2O_5	SO_3	SiO_2	Ins.	Al_2O_3	CaO	MgO	Co_2O_3	Loss.
Tilly Foster, N. Y.[1]	48.82	0.02	0.08	0.21	0.05	. .	11.75	. .	3.50	2.15	13.24	. .	2.30
Port Henry, N. Y.[2]	68.24	0.017	. .	. .	0.038	. .	. . .	4.32	0.28	0.14	. . .	. .	0.972
Crown Point, N. Y.[3]	49.74	0.035	0.02	. .	0.08	. .	27.48	. .	1.97	0.36	0.10	. .	. .
Sussex County, N.J.[4]	50.32	0.01	. .	0.32	0.03	. .	18.27	. .	0.27	5.00	2.73	. .	3.55
Cornwall Mine, Pa.[5]	59.229	0.032	2.910	Trace	0.072	0.039	11.082	CuO 1.840	. .	1.00	1.286	0.20	1.629
Cranberry, N. C.[6]	66.53	Trace	0.25	0.22	Trace	. .	. . .	4.02	1.03	1.06	0.23	. .	1.15

[1] "Mineral Resources of the United States," 1883, p. 273.
[2] *Op. cit.*, 1886, p. 45.
[3] "Mineral Resources of the United States," 1886, p. 46.
[4] *Op. cit.*, 1883, p. 275.
[5] "Transactions American Institute of Mining Engineers," IV, p. 325.
[6] "Mineral Resources of the United States," 1883, p. 27.

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[1] Tenth Census, 1880, XV, p. 30.

[2] Rossi, "Transactions American Institute of Mining Engineers," XXI, p. 832; XXVI, pp. 144, 997; XXX, p. 832; "Iron Age," 1896, LVII, pp. 354, 464; "Engineering and Mining Journal," 1890, July 4, 11; "Universal Industry," IX, p. 715.

(2) *Hematite Ores.* There is quite a variety of hematite ores. Hard hematite (specular, hard specular ore), martite (pseudomorphe of hematite after magnetite), red kidney ore, soft hematite, earthy hematite, red limestone, oölitic hematite (red fossil ore, dyestone, flaxseed ore). The better grades of hematite ores are, as a rule, rich in iron, as they are very free from gangue. They run 58–65 per cent. Fe, low-grade ores 30–50 per cent. Hematite is less difficult of reduction than magnetite; its reducibility varies greatly from the hard specular ores, difficult to reduce, to the earthy or calcareous hematites, comparatively easy to reduce. The gangue is

COMPLETE ANALYSES OF CARGO-SAMPLES OF ORE.

Ore.	Fe	P	SiO_2	Mn	Al_2O_3	CaO	MgO	S	Loss by ignition.	H_2O
	GOGEBIC RANGE, MICHIGAN AND WISCONSIN.									
Ashland	54.29	.040	6.09	.213	2.87	.320	.249	.010	2.047	11.00
Atlantic	55.289	.0428	...	...	...	...	...	...	...	10.65
Aurora	55.2360	.02984	4.1946	.3257	1.655	.2465	.1144	...	2.553	11.952
Colby, Bessemer	57.857	.035	2.447	.233	.917	.143	.188	.005	1.709	10.020
Norrie	56.5662	.03612	3.5620	.3567	1.694	.2853	.1070	...	2.318	10.814
Tilden	55.505	.04318	2.598	.669	1.120	.460	.321	...	2.328	13.11
Winona	51.036	.041	7.152	1.948	.804	.187	.080	.005	1.948	10.600
	MARQUETTE RANGE, MICHIGAN.									
Angeline, hard	63.65	.010	...	...	...	...	...	...	...	5.57
Angeline, hematite	57.85	.036	...	...	...	...	...	...	...	11.17
Champion No. 1, crushed	63.49	[illegible]	4.51	.198	2.36	.317	.288	.0129	...	.80
Champion, hematite	47.76	.363	8.99	.256	1.53	2.89	1.65	.048	5.80	8.60
Cliffs Shaft, crushed	61.24	.0969	4.43	.237	2.166	1.266	.959	.0207	.623	1.06
Cliffs Shaft, lump	63.22	.0866	3.286	.248	1.872	1.224	.836	.0189	.557	.42
Jackson, South	38.9428	.0751	27.1225	2.5840	1.5119	.1283	.0641	.0165	3.2071	8.37
Lake	51.90	.1086	4.981	.425	2.451	.434	.469	.0113	3.225	13.07
Michigamme	58.45	.102	11.54	.187	2.20	.829	.997	.024	.148	1.26
Negaunee	52.815	.0549	6.55	.310	2.552	.877	.5387	.0593	1.754	11.383
Republic, crushed	64.84	.0455	3.960	...	...	...	...	...	...	1.00
Republic, specular	66.74	.0396	2.286	Trace	.7395	.217	.148	.0247	.227	1.00
Salisbury	52.56	.097	6.089	.252	2.009	.487	.487	.0104	1.931	13.01
Tilden, silica	41.35	.030	37.42	.058	.815	[illegible]	.098	.0098	.265	1.78

Complete Analyses of Cargo-Samples of Ore. — *Continued.*

Ore.	Fe	P	SiO_2	Mn	Al_2O_3	CaO	MgO	S	Loss by ignition.	H_2O
Menominee Range, Michigan and Wisconsin.										
Chapin	55.368	.0574	5.835	.3850	1.092	1.251	2.679	.0037	2.380	6.63
Crystal Falls	53.606	.705	3.902	.347	1.663	2.376	.968	.005	2.010	8.600
Pewabic	59.3491	.0110	4.3795	.1199	.9128	.5255	1.1248	.0009	.9497	7.80
Quinnesec, non-Bessemer	59.002	.106	3.889	.199	.664	.189	.246	.005	2.067	5.140
Quinnesec, siliceous Bessemer	43.823	.031	29.443	.117	1.066	.254	.371	.003	2.660	2.180
Mesabi Range, Minnesota.										
Adams	56.719	.0307	3.154	.6381	.976	.1337	.0723	.0587	4.130	9.61
Biwabik	58.64	.035	2.86	.415	.876	.175	.10	.008	3.81	7.72
Clark	56.148	.0303	4.182	.929	.858	.205	.160	.0044	. . .	10.62
Elba	55.7891	.03526	3.4721	.8409	.8047	.0904	.1537	.0090	4.7018	9.58
Fayal	57.860	.0335	2.675	.589	.680	.362	.136	.0036	3.346	9.31
Genoa	56.453	.0287	4.147	.412	.691	.395	.197	.0062	3.088	10.22
Mahoning	57.93	.042	2.19	.349	1.308	.12	.107	.007	2.66	10.40
Mountain	55.833	.03632	3.457	.1917	1.707	.0927	.0612	.0122	2.057	12.459
Sauntry No. 1	56.221	.0483	5.414	.455	1.713	.191	.100	.0054	. . .	8.85
Stevenson	59.504	.033	2.769	.284	.660	.238	.165	.004	2.494	8.300
Vermilion Range, Minnesota.										
Chandler	61.224	.0380	4.001	.095	1.891	.361	.190	Trace	.931	4.96
Pioneer	59.108	.0374	5.301	.1226	2.388	.0693	.0159	. . .	1.264	6.34
Savoy	59.532	.0476	3.230	.0746	2.222	.2315	. . .	. . .	2.502	6.63
Sibley	60.160	.0451	3.760	. . .	. . .	. . .	. . .	. . .	. . .	6.00
Vermilion	64.205	.1345	4.195	.059	1.335	.643	.405	.0148	.583	1.04
Michipicoten Range.										
Helen	55.99	.106	5.10	.12	.86	.12	.08	.0598	9.49	5.00

liable to be more siliceous than with magnetites, at least with the better grades of ores. There is, however, such a diversity in the gangue of the different hematites that no general characteristic can be given. Of the European deposits those of Elba, Spain, and Algiers have gained a wide reputation. Much Cuban ore is imported to the works on the Atlantic coast. The deposits of hard and soft hematite on the northwest and southwest shores of Lake Superior are unique both in quality and quantity. Red fossil ores and red limestones are abundant in the South, especially in Alabama and Tennessee. Hematite forms the greater proportion of the country's product of iron ore.

P. — The deposits of Lake Superior are large and rich in iron (58 per cent. Fe and over) and have a siliceous gangue. As a rule they run low in P, S, and Mn. The deposits of the South are not as rich in Fe, and run high in P (non-Bessemer ores). Of the hematite mined in 1880[1] 60 per cent. was Bessemer ore.

S is generally low in hematite, although FeS_2 is occasionally found. The lake ores contain the least S, then come Missouri ores; southern ores are highest.

Ti is rare.

In the "Iron Trade Review" of July 13, 1903, are given average analyses of the ores of the above six ranges, with highest and lowest percentages of constituents, and other tables in which the ranges are grouped according to the percentages of Fe, Al_2O_3, bases, SiO_2, H_2O, P, S, Mn and loss on ignition of their ores.

(3) *Limonite* (*Brown Hematite*) *Ores.* According to the compactness, structure, and varying percentage of H_2O, a number of varieties are distinguished: compact brown hematite, earthy (ochreous) brown ore, buhrstone (ferriferous limestone), oölitic brown ore, bog or lake ore. The best grades contain 60 per cent. Fe, the worst say 20 per cent. Fe. The ore is generally very much contaminated with gangue and has to be hand-sorted, and often concentrated by mechanical means (log-washers). Limonite is very easily reduced, as upon heating it gives up its combined water, becomes porous, and thus offers a large surface to the reducing gases. The character of the gangue varies greatly, as limonites occur in nearly every formation. Limonites are found in many parts of the United States, but especially in the most eastern valley of the Appalachian Range, extending from Canada to Alabama; also in a belt extending from Alabama to western Missouri, and in western Pennsylvania and Ohio.

Mn occurs very frequently with limonite; in fact it is almost always present.

P occurs in most limonites. Of all the ore mined in the United States in 1880[2] only 8 per cent. was within the Bessemer limit.

S is also present to considerable extent.

Ti is not found.

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[1] Tenth Census, 1880, XV, p. 29.

[2] *Op. cit.*, 1880, XV, p. 29.

ANALYSES OF LIMONITES.

Element.	PENNSYLVANIA.			ALABAMA.		TENNESSEE.		GEORGIA.
	United States Census, 1880, XV,			Trans. Am. Instit. Min. Eng., XV, p. 200.	Trans. Am. Instit. Min. Eng., XII, p. 166.	Trans. Am. Instit. Min. Eng., XV, p. 208.	Trans. Am. Instit. Min. Eng., XV, p. 198.	Min. Res. U. S., 1886, p. 95.
	p. 181,	p. 183,	p. 198.					
Fe	34.950	48.013	40.580	51.24	32.284	50.86	36.20	57.84
P	0.509	0.149	0.036	0.164	0.380	0.472	0.29	0.24
S	0.017	0.025	0.370	. . .	0.070	0.406	. . .	None
Mn	5.123	Trace	Trace	0.770	0.201	. . .	4.81	0.71
P_2O_5	1.169	0.340	0.082	. . .	0.870	. . .	. . .	0.54
SO_3	0.042	0.060	0.925	. . .	. . .	. . .	. . .	. . .
SiO_2	. . .	. . .	. . .	11.03	. . .	10.98	16.64	4.37
Ins.	26.700	9.930	24.538	. . .	4.620	. . .	. . .	. . .
Al_2O_3	3.053	2.010	2.100	8.43	1.850	. . .	. . .	0.13
CaO	0.110	0.270	0.120	0.82	0.390	. . .	7.01	0.05
MgO	0.418	1.480	0.115		0.750	. . .	. . .	0.03
Co_2O_3	0.140	0.040	. . .	. . .	. . .	. . .	. . .	. . .
Loss by ignition	11.384	17.700	. . .	4.92	7.510	. . .	11.11	11.96

(4) *Siderite Ores.* According to the predominance of siderite or of impurities, such as argillaceous and carbonaceous matter, the following varieties are distinguished: spathic ore (granular or compact pure siderite, called spherosiderite when in concretionary form, 45 per cent. Fe), clay ironstone (argillaceous siderite, 38–40 per cent. Fe), blackband ironstone (bituminous clay ironstone, with 10–15 per cent. bitumen, 25–50 per cent. Fe). The ores contain 25–45 per cent. Fe. In spathic ore the FeO is often replaced in part by MnO, CaO, MgO (ankerite). As a rule it runs low in P and S. Clay ironstone and blackband ores are liable to run high in P. Carbonates are not so easy of reduction as might be expected, as the FeO is liable to be slagged. By roasting they are oxidized, and having become also porous they are reduced without difficulty. The nature of the gangue differs with the three varieties. Spathic ores are more earthy, argillaceous ores are rich in SiO_2, and blackband ores stand between the two.

Mn occurs abundantly with spathic ores (up to 20 per cent.); argillaceous ores contain up to 7 per cent. Mn; blackbands have less.

P is low in spathic ores, high in argillaceous ores, lower in blackbands, but they are rarely within the Bessemer limit.

S is high in all three varieties.

Ti does not occur.

Siderite ores are extensively mined in Europe. They are of little importance in the United States. Spathic ores occur in Connecticut, Vermont, and New York; spathic ores partly converted into argillaceous and blackband ore, and limonite in Alabama, Kentucky, Maryland, Ohio, Pennsylvania, West Virginia.

ANALYSES OF SIDERITE ORES.

		Fe	P	S	Mn	P_2O_5	SO_3	SiO_2	Ins.	Al_2O_3	CaO	MgO	Loss by ignition.
Spathic ores.	New York [1] . .	38.80	0.19	0.47	1.110	0.430	. .	. .	14.77	2.13	1.33	2.61	24.10
	Pennsylvania [2] .	38.80	0.195	0.408	0.619	0.440	1.020	. .	10.605	1.092	1.240	1.203	33.549
	Pennsylvania [3] .	38.750	0.121	0.262	0.994	0.277	Trace	. .	2.920	0.528	5.650	2.396	36.820
	Pennsylvania [4] .	36.500	0.148	0.148	0.677	0.338	0.040	. .	12.403	3.220	1.210	1.690	31.780
Clay iron-stone.	Tennessee [5] . .	38.700	0.574	0.192	0.033	1.314	0.133	. .	6.410	2.240	4.536	0.569	33.378
	Alabama [6] . . .	35.000	0.149	0.137	0.118	. .	. .	5.21	. .	4.050	5.91		2.40
Blackband ironstone.	Ohio [7]	30.150	0.592	0.175	1.10	1.362	. .	. .	13.26	7.93	2.07	0.65	30.83
	Ohio [8]	27.320	0.043	0.170	2.10	0.096	. .	25.52	. .	9.50	0.84	1.51	33.17
	Ohio [9]	22.62	0.018	0.300	1.85	0.032	. .	24.16	. .	5.75	. .	. .	20.06

[1] "Transactions American Institute of Mining Engineers," IV, p. 341.
[2] Census, 1880, XV, p. 204.
[3] *Op. cit.*, p. 207.
[4] *Op. cit.*, p. 209.
[5] *Op. cit.*, p. 203.
[6] "Mineral Resources," 1886, p. 87.
[7] *Op. cit.*, 1886, p. 60.
[8] *Op. cit.*, 1886, p. 61.
[9] *Ibid.*

(5) ***Purple Ore and Blue Billy.*** Purple ore is a ferric oxide obtained from dead-roasting an iron sulphide, generally FeS_2, in the manufacture of H_2SO_4. If it has been further treated in the wet way to extract the copper (AgAu) it is called blue billy. This ore contains 96 per cent. Fe_2O_3, with a little Pb, Cu, S, etc. It is used as iron ore in the blast furnaces, as a decarburizer in making malleable castings, and as fettling in puddling furnaces.

ANALYSES.

	Fe_2O_3	Al_2O_3	Cu	S	P	$PbSO_4$	$CaSO_4$	CaO	MgO and Alk.	Na_2SO_4	NaCl	H_2SO_4	Ins.
England [1]	90.61	. .	0.15	0.08	. .	1.46	0.37	. .	. . .	0.37	0.28	. .	6.30
	95.10	. .	0.18	0.07	. .	1.29	0.49	. .	. . .	0.29	. .	. .	2.13
Oker [2] . . .	79.00	3	. .	. .	. .	. .	. .	2.5	1.00	. . .	. .	5.5	6.00

[1] Lunge, "Sulphuric Acid and Alkali," 1891, I, p. 809.
[2] Bräuning, "Zeitschrift für Berg-, Hütten- und Salinen-Wesen in Preussen," 1877, XXV, p. 162.

(6) *Franklinite Ores.* They are magnetite ores in which part of the Fe has been replaced by Zn and Mn. They occur in Franklin and Sterling, Sussex County, New Jersey. The ores are first worked for their zinc content, and leave behind a clinker called ***residuum***, which forms the raw material for the production of spiegeleisen (pig iron with 10–20 per cent. Mn) in the blast furnace.

ANALYSES OF RESIDUUM.[1]

	SiO_2	Fe_2O_3	Al_2O_3	MnO	CaO	MgO	ZnO	P	Fe	Mn
New Jersey Zinc Company . .	19.97	33.21	2.25	17.83	11.96	2.30	10.74	0.037	23.25	13.82
	25.02	31.06	6.36	16.22	10.73	2.67	6.98	. . .	21.74	12.56
	23.47	33.84	8.24	15.66	11.04	1.84	4.98	. . .	23.69	12.13
Passaic Zinc Company	18.56	36.21	4.07	15.69	16.70	1.92	4.70	C 1.85	23.35	10.74
Lehigh Zinc and Iron Company .	25.16	28.99	6.57	9.45	12.80	4.55	4.55	0.036	20.39	7.32

(7) *Metallurgical Products Rich in Iron.* In the manufacture of wrought iron is produced puddle cinder, essentially a basic ferrous silicate intermixed with Fe_2O_3. It contains SiO_2, 15–20 per cent.; FeO, 50–70 per cent.; Fe_2O_3, 10–20 per cent., and is rich in P. In the manufacture of steel, slags rich in iron are formed which are charged back into the blast furnace. In bringing blooms or billets to the welding heat before they are rolled or hammered some of the iron is oxidized, scales off, and combines with the siliceous hearth material to a slag, mill, or heating cinder, running say 30 per cent. Fe, which also goes to the blast furnace.

ANALYSES OF METALLURGICAL PRODUCTS RICH IN IRON.

J. WHITING, ILLINOIS STEEL COMPANY, NORTH WORKS.

Material.	Fe	P	SiO_2	Mn	Al_2O_3	CaO	
Mill cinder . . .	53.76	.675	23.00	.58	1.80	.40	From E. C. I. and S. Company.
Roll scale . . .	72.72	.057	1.90	. . .	. . .	. . .	From North Works, Mill.
Mill cinder . . .	52.53	.060	28.10	.68	1.72	. . .	From North Works, Mill.
Roll scale . . .	69.86	.204	6.51	. . .	. . .	. . .	From Western Tube Company.

(8) *Harmful Minerals* occur more or less distributed through the various iron ores; if in larger quantities they have to be removed by preliminary mechanical or chemical operations; if in sufficiently small amounts they can often be neglected.

(9) *Minerals Used for Making Special Cast Irons and Steels: Mn, Cr, W. Manganese Minerals* are mainly oxides. They have been given above. In commerce manganese ores are divided into two classes: manganese ores and manganiferous iron ores. The dividing line is 70 per cent. MnO_2 (= 44.25 per cent. Mn). The first class is used for chemical purposes;

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[1] "Engineering and Mining Journal," March 20, 1886.

that is, the oxidizing power of MnO_2 is utilized: in the second class the Mn of the mineral enters into combination with iron to form spiegeleisen (up to 20 per cent. Mn) and ferromanganese (up to 80 per cent. Mn). Manganese ores occur with limonites and also with carbonates, rarely with red hematites, and not at all with magnetites. In the United States manganese ores occur in the limonite belt extending from Maine to Alabama, in Arkansas and in California. Manganese ores are very difficult of reduction; generally a part of the Mn goes into the slag and makes this very thin and fluid. Manganese oxides are sometimes roasted before being smelted.

P occurs quite frequently in manganese ores, and ores low in P have a very great value.

S.—The presence of S is not of great importance if the ore is to be roasted.

Ti does not occur with Mn ores.

ANALYSES OF MANGANESE ORES.

J. WHITING.

Material.	Fe	P	SiO_2	Mn	Al_2O_3	CaO	MgO	BaO	Moisture.
Chilian	3.33	.090	12.70	41.72	...	...	...	...	1.40
Chilian	.66	.021	9.76	45.13	...	...	...	...	3.47
Keystone	3.58	.158	3.33	51.83	3.13	.80	.32	.52	5.91
Abbott	4.83	.401	3.92	50.57	...	...	...	...	4.25
Clinch	5.00	.095	18.08	46.35	...	...	...	...	3.50
McDowell	12.23	.373	16.08	35.18	...	...	...	...	6.63

Chromite (49–52 per cent. Cr_2O_3) comes principally from Greece, Turkey, Quebec, and Newfoundland. Some ore is mined in California.

Wolframite (75 per cent. WO_3) is concentrated to 50–70 per cent. ore. It is mined in Arizona, New Mexico, South Dakota. Most of the ore comes from Bohemia, Saxony, Cornwall, Spain, Portugal, Australia, and New Zealand.

SECTION 5. **Preparation of Iron Ores.** The aim in preparing iron ores is to bring them to a suitable size, to make them porous, and to remove impurities by mechanical or chemical means.

(1) *Crushing and Breaking of Iron Ores.* An ore passing through the blast furnace will expose more surfaces to the reducing action of the ascending gases the smaller the single pieces; but if they are too small they will pack and hinder the passage of the gases. The right size will depend upon the height of the furnace and the reducibility of the ore. Low (charcoal) furnaces can stand smaller-sized ore than high (coke) furnaces. Common sizes are fist- to pea-size. An ore difficult to reduce must be crushed finer than that which is readily deoxidized. Of the difficultly-reducible iron minerals silicates of iron are the most refractory; then follow Fe_3O_4, $Fe_2O_3 + H_2O$, $FeCO_3$, the hard varieties being less easily reduced than those that are soft. Ores are broken in crushers, rolls, etc. If ores are very fine (Mesabi ores) they cannot be easily smelted alone, although it is being done.[1] Usually they are mixed with coarser ores;

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[1] Uehling, Barrows, "Transactions American Institute of Mining Engineers," 1904.

occasionally they are compressed by machinery into briquettes or eggettes, the bond used being lime, rosin, tar, pitch, etc. Flue-dust (particles of fine ore carried away by the gas current and deposited in suitable chambers) is frequently briquetted. Edison[1] briquetted magnetite concentrated with a resinous binder at the Ogden Mine, New Jersey. They contain Fe, 67–68; SiO_2, 2–3; Al_2O_3, 0.4–0.8; Mn, 0.05–0.10; CaOMgO, trace; P, 0.028–0.033; resinous bond, 0.78.

(2) *Calcination* (*Burning*) *of Iron Ores.* The object of calcining or burning an ore is to loosen its texture or to drive off volatile matter, such as H_2O or CO_2. Ores are rarely calcined at present. The apparatus is similar to that used in roasting.

(3) *Concentration of Iron Ores* is the process of removing impurities by mechanical means. It involves crushing to set free impurities, sizing to obtain particles of approximately even cubical contents, so that the heavy iron mineral may be freed from the lighter impurity by washing or magnetic separation.[2] Ores that are not readily attracted by the magnetic have been made so by a preliminary roasting.

(4) *Roasting of Iron Ores. In General.* Roasting an iron ore means heating it with access of air to a temperature that generally precludes fusion, in order to effect a chemical change of the iron mineral ($FeCO_3$), of the impurity (FeS_2), or of both; incidentally the ore undergoes a physical change, becoming porous.

Magnetites, when smelted with charcoal as in Sweden, are, as a rule, roasted before smelting, in order to make them porous and to peroxidize the iron to some extent; they lose 3–5 per cent. in weight by giving up water of crystallization or hygroscopical H_2O, but gain somewhat by taking up oxygen.

Hematites are roasted only when too rich in S. Occasionally they are calcined to make them porous. The loss in weight amounts to 3–5 per cent.

Limonites were formerly more frequently calcined than at present. As they are likely to be rich in S they may have to be roasted. They lose 8–20 per cent. in weight.

Siderites are generally roasted previous to smelting to drive off CO_2, H_2O, C_xH_y, and S, and to bring the FeO to a higher state of oxidation varying from Fe_6O_7 to Fe_2O_3. According to Wedding[3] the following reactions take place upon heating $FeCO_3$:

without access of air, $6FeCO_3 = Fe_6O_7 + CO + 5CO_2$;
with access of air, $FeCO_3 + O = Fe_2O_3 + 2CO_2$.

Spathic Ores when pure lose 40 per cent. in weight; commonly, however, only 28–35 per cent.; clay ironstones 18–30 per cent., blackbands 36–44 per cent.[4]

Manganese Ores are frequently calcined to drive off some of the oxygen: $3MnO_2 = Mn_3O_4 + O_2$.

Roasted ore exposed for any length of time to moist air absorbs from 2 to 6 per cent. H_2C and CO_2.

For the behavior of metallic sulphides in an oxidizing roast consult Hofman, "Metallurgy of Lead," 1899, page 150.

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[1] "Iron Age," LX, No. 18, p. 1.

[2] See Richards, "Ore Dressing."

[3] "Eisenhüttenkunde," II, p. 450.

[4] "Transactions American Institute of Mining Engineers," XXII, p. 107; XXV, p. 203.

GJERS KILN.

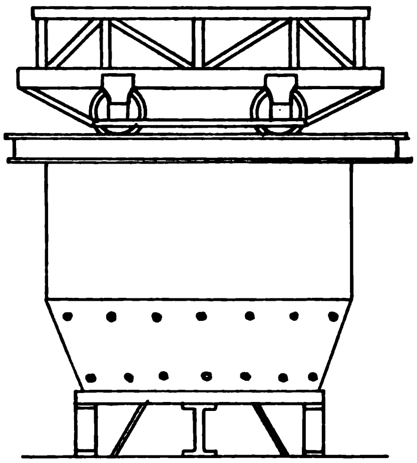


FIG. 25.

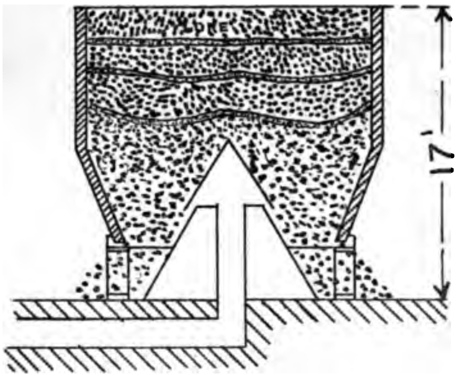


FIG. 26.

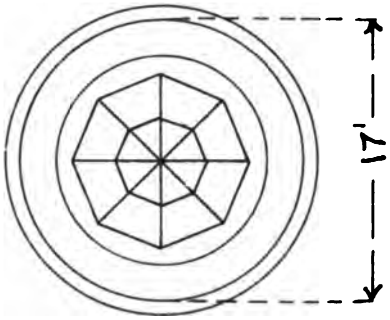


FIG. 27.

DAVIS-COLBY KILN.

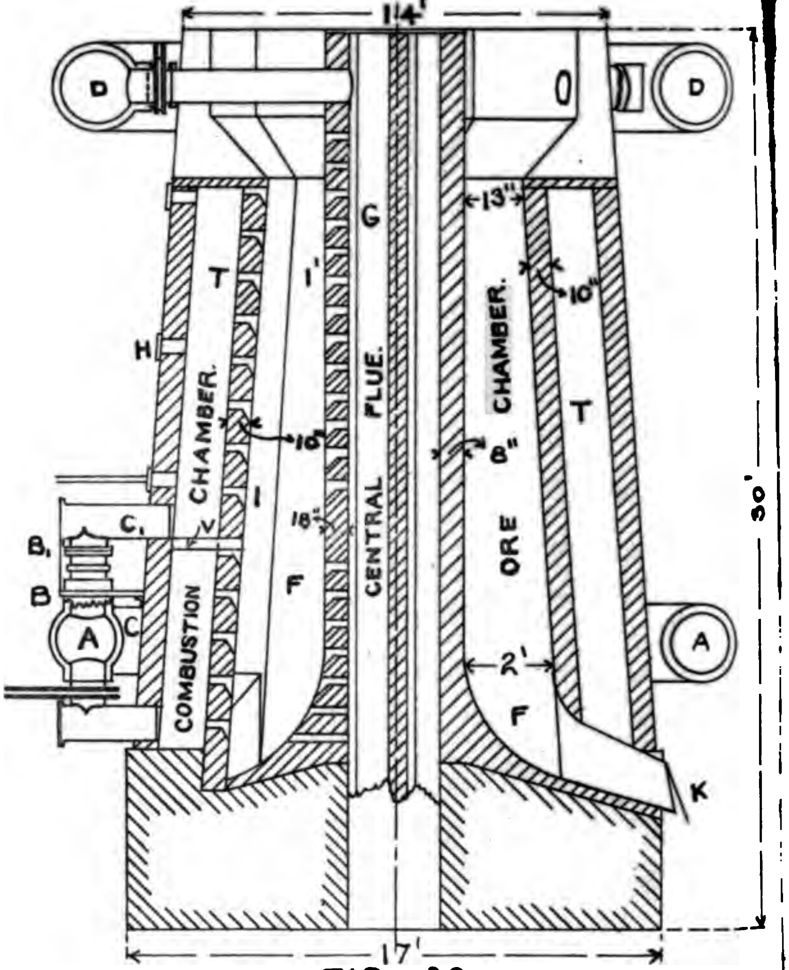
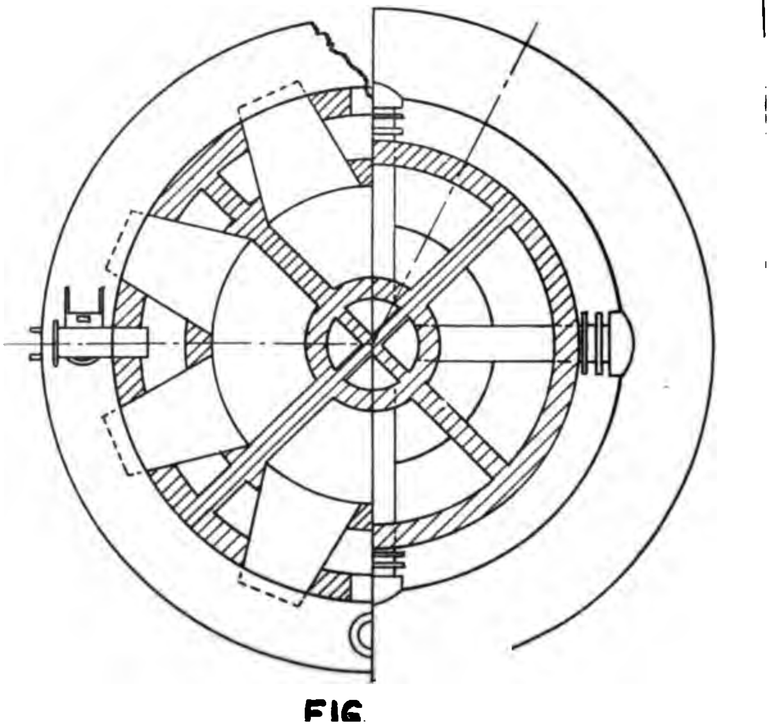


FIG. 28.



FIG

In roasting an ore the time given, the temperature maintained, and the size of particles have to be varied with the character of the ores. Peroxidation and desulphurization require time, and as a rule a low temperature (dark to light red). Occasionally, *e.g.*, with Swedish magnetites, which have to be completely desulphurized (as they are smelted in charcoal iron blast furnaces with an acid slag, precluding the use of lime for the elimination of S), the heat may have to be raised to incipient caking. An ore containing $FeCO_3$ has to be roasted at a lower temperature than one containing Fe_2O_3 or Fe_3O_4, as the former is readily scorified by an acid gangue. Pure ores can be coarser than when impure. An ordinary size is 1–2 cubic inches. Fine ores are mixed with coarse ores in roasting; when there are too many fines, special fine-ore furnaces have to be used. A well-roasted ore should be somewhat friable and reddish in color, excepting sintered magnetite. The more uniform the lump the better the roast.

(5) *Roasting Apparatus.* Iron ores are roasted in heaps, stalls, kilns, and reverberatory furnaces.

(*a*) *Roasting in Heaps* is a primitive method of operating, and little used today. Ore and fuel are placed in alternate layers upon a bed of wood or coal, the thickness of the layer of wood decreasing with the height of the heap. Blackbands containing 25–30 per cent. combustible matter required no addition of fuel. The heap has the form of a truncated pyramid, is 12–20 feet wide (blackbands 30 feet), 100–200 feet long, 3–17 feet (generally 9 feet) high.[1] Horizontal and vertical flues are often left open to assist in kindling the heap and in regulating the draught; the temperature is regulated by increasing or decreasing the thickness of the layer of fine ore with which the sides are covered. A heap burns 2–4 weeks, requires 10 per cent. fine coal (mine-fines) on the weight of the roasted ore. Fulton[2] gives the cost of heap-roasting sulphurous magnetite at Cornwall, Pennsylvania, in 1882 at $0.558 per gross ton of roasted ore, viz., labor, burning, $0.218; wood and hauling, $0.07; 15 per cent. coal and placing, $0.15; loading on cars, $0.12; total, $0.558.

(*b*) *Roasting in Stalls.* Fireplaces enclosed by three permanent walls are hardly found at all today.[3]

(*c*) *Roasting in Kilns*[4] is the method universally used when the ores are not too fine. The kilns used are shaft-like structures. The roasted ore that has been charged at the throat is drawn at intervals at the base, thus making the process a continuous one. The ore moves slowly downward through the shaft, which is constantly refilled; the gases ascend from the base, gradually giving up their heat to the descending charge, thus preparing it for the hot oxidizing zone near the base of the furnace. The advantages of this roasting over roasting in heaps are that the process is a continuous one, the consumption of fuel is smaller, the temperature is more even, the roast is more uniform, and less labor is required. Both solid and gaseous fuels are used. The solid fuels are the fines of the fuel charged with the ore into the blast furnace; that is, charcoal, coke, anthracite, sometimes, also, non-caking bituminous coal. The fuel is usually charged in alternate layers with the ore at the throat. It is first

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[1] "Transactions Institution of Mining Engineers," XXII, p. 107; XXV, p. 203.

[2] "Transactions American Institute of Mining Engineers," XII, p. 367.

[3] Example, "Transactions American Institute of Mining Engineers," VIII, p. 517.

[4] See Pavloff, Atlas, Plates 1–4.

warmed and then ignited by the ascending current. The gaseous fuels are the waste gases of the blast furnace or producer gas. The gas is introduced at or near the base of the furnace through special nozzles; the air necessary for combustion is admitted through the openings for the withdrawal of the charge at the base, or higher up through the stoking holes, or between these two through special air flues. Gaseous fuel has the advantage over solid fuel in that a uniformly strong oxidizing flame can be had; the height of the temperature is easily regulated; and the ore does not become contaminated with the ash of the fuel. In some cases the solid fuel is burned on grates placed outside of the furnace and the flame conducted into the furnace at the base. This method has the same advantage as the gaseous fuel, that fuel ashes do not come in contact with the ore; but it is expensive, as it requires long-flamed fuel, extra labor, and the heat is liable to be excessive. The form and height of a kiln vary somewhat with the character of the ore to be treated. Kilns are generally circular in horizontal section. The vertical section may show a tapering toward the throat if the ores are to move quickly, a taper toward the discharge if slowly. Frequently the furnace has parallel or slightly tapering walls which narrow down suddenly toward the bottom (have a bosh). Here the ores will lie compactly near the base and move slowly, thus being exposed for a long time to oxidizing influences. The height varies with the size and fusibility of the ore and the amount of care that is to be given to the operation. Large lumps and sulphurous ores go into high furnaces; fusible ores into low furnaces.

I. KILNS USING SOLID FUEL. (1) *The Gjers Kiln, Colebrook Furnaces, Lebanon, Pennsylvania* (Figures 25–27). This kiln,[1] serving to roast magnetite ore, is a modification of the original Gjers,[2] intended to roast clay ironstone of Cleveland, England. It has a circular iron casing with bosh (17 and 15½ feet in diameter, and 17 and 18½ feet high) lined with brick, which rests upon iron plates supported by eight cast-iron columns 3 feet high. Ore and fuel are charged in alternate layers at the top; the roasted ore is drawn from the bottom; the air enters at bottom through the hot roasted ore, under the distributing cone, and at the side through the draught holes, which also serve as poking holes. The kiln roasts in twenty-four hours 10–50 tons ore, the amount increasing with the proportion of lump ore charged. It requires 50–100 pounds coal per ton of ore. The ore remains two to ten days in the furnace. The cost is about thirty cents per ton of roasted ore delivered into furnace-charging buggies; twelve cents go for fuel, eighteen cents for labor.

(2) *The Grittinger Roaster*,[3] also in use at Cornwall, Pennsylvania, is similar in construction. It is, however, raised and so arranged that the roasted ore can be drawn into buggies, thus reducing the labor of handling. The diameter is 15 and 12 feet, the height 12 feet. It holds 125 tons ore and roasts daily 18–25 tons. Ninety pounds buckwheat anthracite are burned per ton of ore. Valentine[4] gives the following data of roasting Cornwall ore with 2.5 per cent. to 3.5 per cent. S:—

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[1] "Transactions American Institute of Mining Engineers," XII, p. 374.

[2] "Journal Iron and Steel Institute," 1871, II, p. 202.

[3] "Transactions American Institute of Mining Engineers," XII, p. 377.

[4] "Transactions American Institute of Mining Engineers," XVIII, p. 309.

Per cent. total sulphur.	Per cent. sulphate sulphur.	Per cent. sulphide sulphur.	Per cent. of total sulphur as sulphate sulphur.
1.333	0.283	0.850	24.977
1.380	0.135	1.245	9.782
1.873[1]	0.096	1.777	5.125

[1] Clinkered.

(3) *Kiln of the Umsconetcong Iron Works, New Jersey.*[1] This is a circular shaft furnace tapering toward the throat, 15 and 13 feet in diameter, 12 feet high (15 would be preferable), having 3-foot supporting columns. It treats in twenty-four hours 40–50 tons magnetite, reducing the S from 0.883 per cent. to 0.39 per cent., and requires 60 pounds fuel per ton roasted ore.

II. KILNS USING GASEOUS FUEL. (1) *The Westman Kiln.*[2] This was first built in Sweden (1851) for roasting magnetite and heating it to the point of sintering in order to eliminate as much S as possible. It is conical, 30 feet high, 10 and 6 feet in diameter. Waste gas from a charcoal iron blast furnace is admitted near the bottom; air enters through the discharge ports, being first warmed by passing through the cooling ore. There are poking and peep holes; 40 tons of ore are roasted in twenty-four hours. Rinman[3] gives 500 cubic feet charcoal blast-furnace gas as being required for roasting 1 ton of ore. This would correspond to 333 cubic feet of producer gas from bituminous coal, for which would be required about 130 pounds coal.

(2) *The Langdon Ore-roasting Furnace*[4] was one of the first gas-roasting furnaces aiming to withdraw the ore from the influence of the flame, after it had been heated sufficiently to give off its sulphur and had become porous, and then to let it cool slowly in a current of warm air. It is out of use now.

(3) *The Improved Davis-Colby Ore Kiln*[5] (Figures 28–29). *G*, the central flue, is divided by radial walls into four parts. Each is connected by a horizontal pipe with the annular draft-flue, *D*, leading to the stack. *F*, the annular ore chamber, is also divided into four parts to prevent the ore from rolling and to permit a proper regulation of draught. The ore is dumped from mine cars into a hopper at the throat, or charged by a conveyor and drawn at the bottom through eight chutes, *K*. *T* is the combustion chamber. It receives blast furnace or producer gas through the annular pipe, *A*. The gas is then admitted to the lower or upper division by the pipes *C* and *C'*, and regulated by the valves *B* and *B'*. Air for combustion enters through *H*. The products of combustion pass through *I* into the ore chambers, calcine or roast the ore, and thence through *I'* into the central flue. The partition *L* is not necessary in an ordinary oxidizing roast. It was introduced by Phillips for a reducing roast, the aim being to render magnetic Alabama red fossil ore and then enrich it by concentration (see below). At Shelby,

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[1] "Transactions American Institute of Mining Engineers," XV, p. 678.

[2] "Journal Charcoal Iron Workers," II, p. 208; Åkerman-Turley, "Das Rösten der Eisenerze," Leipsic, 1880, p. 48; Ledebur, "Eisenhüttenkunde," 4th ed., p. 261.

[3] "Berg- und Hüttenmännische Zeitung," 1872, p. 112.

[4] "Transactions American Institute of Mining Engineers," IX, p. 304.

[5] "American Manufacturer," May 5, 1899.

Alabama,[1] one furnace, 36 feet high, 17 feet in diameter at bottom and 14 feet at top, holding 110 tons ore, heated with gas from a Taylor producer, treats in twenty-four hours 125 tons red fossil ore; requires $2\frac{1}{2}$–3 tons bituminous coal and four men. The coal has the following composition: H_2O, 1.00; V. M., 37.00; F. C, 59.40; ash, 2.00; total, 100.00; S, 0.90, and gives about 130,000 cubic feet, or 8,000 pounds gas per ton. In a reducing roast Phillips[2] used 3,545 cubic feet gas (CO, 25; CO_2, 6; H, 13; O, 0.40) per ton of ore, and put through 110 tons in twenty-four hours. The roasted ore contained 45 per cent. Fe and 30 per cent. SiO_2. After having been passed over a Hoffman magnetic separator[3] the heads contained 58.9 per cent. Fe and 11.5 per cent. SiO_2. One ton of concentrates can be produced from 3 tons of ores at a cost of \$1.50.[4]

ANALYSES OF ROASTED ORES.

	MAGNETITE.		LIMONITE.			SIDERITE.	
	Cornwall, Pennsylvania.[1]		Shelby Iron Works, Alabama.[2]		Hocking Valley, Ohio.[3]	Blackband, Tuscarawas Valley, Ohio.[4]	New York.[5]
Fe	55.90	55.43	60.01	53.96	44.65	52.50	54.042
Mn	. . .	. . .	. . .	. . .	. . .	1.65	2.165
P	. . .	. . .	0.059	0.165	0.175	. . .	0.120
S	0.38	01.7	0.018	0.075	0.40	Trace	0.224
Al_2O_3	3.80	32.4	1.93	5.34	9.31	0.60	2.768
CaO	2.30	20.6	0.52	0.65	1.120	2.80	1.650
MgO	3.52	34.7			0.42	1.48	1.167
K_2O	. . .	. . .	. . .	. . .	. . .	. . .	. . .
P_2O_5	. . .	. . .	0.076	0.212	0.41	0.775	0.275
SO_3	0.97	0.42	0.045	0.187	1.00	Trace	0.560
SO_2	11.85	11.63	9.16	12.85	. . .	. . .	. . .
Ins.	. . .	. . .	. . .	. . .	22.15	17.02	. . .
Loss by ignition .	. . .	. . .	0.65H_2O	0.62H_2O	. . .	0.25H_2O	5.684

[1] "Transactions American Institute of Mining Engineers," XIV, p. 895.
[2] *Op. cit.*, XV, p. 200.
[3] "Mineral Resources of the United States," 1886, p. 59.
[4] *Op. cit.*, 1886, p. 60.
[5] "Transactions American Institute of Mining Engineers," XII, p. 92.

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[1] "Engineering and Mining Journal," December 30, 1893.

[2] "Transactions American Institute of Mining Engineers," XXV, p. 399.

[3] "Engineering and Mining Journal," LII, 1891, p. 680; *op. cit.*, LIII, 1892, p. 662; "Transactions American Institute of Mining Engineers," XVI, p. 602; *op. cit.*, XXV, p. 412.

[4] Results with Cornwall ore. "Transactions American Institute of Mining Engineers," XVIII, p. 311.

(*d*) *Roasting in Reverberatory Furnaces.* Iron ores will be roasted in reverberatory furnaces only when the ore is too fine to be roasted in a kiln. Reverberatory furnaces are not used in this country. If there were a demand for them the mechanical roasters used in treating Cu, and Pb, Zn, Ag, Au ores would be chosen for the purpose. Examples of roasting furnaces for carbonate ore used in Southern France are described by Gromier in the "Bulletin de L'industrie Minérale," 1893, page 465.

(6) *Weathering of Ores.* The object of weathering of ores is to disintegrate them, or to sulphatize metallic sulphides, which, excepting $PbSO_4$, being soluble, are more or less leached out by the rain. Clay ironstone from South Wales, England, is associated with shaly matter. This is disintegrated by exposure to the weather before the iron ore, and can thus be readily separated by cobbing and picking. Weathering is rarely employed at present, as iron plants consume such large amounts of ore as to make the slow process of weathering on the whole impractical.[1]

SECTION 6. **Fluxes.** The slag formed in smelting iron ores is a silicate of Al_2O_3 and CaO. As there is usually a lack of CaO in the ores this is supplied by limestone, $CaCO_3$, incidentally by dolomite, $(CaMg)CO_3$.[2] The limestone should be free from P and S and run low in SiO_2. As a flux dolomite works as well as limestone. The fusibility curves of silicates of lime and of lime-magnesia by Åkerman (Section 22) show that the melting-points are lowered by having lime-magnesia instead of lime alone. In an iron blast-furnace slag consisting of SiO_2, Al_2O_3, and CaO, the replacing of CaO by MgO raises the melting-point. For the removal of S in the form of an alkaline earth sulphide limestone is preferable.[3] Occasionally there is a lack of Al_2O_3 or SiO_2, which is made up by some aluminous silicates, such as slate, schist, granite, or low-grade siliceous iron ores with 50–70 SiO_2, 15–20 Al_2O_3, 2–15 Fe_xO_y, 2–10 CaO, MgO, 8–10 alkali, etc. In only very rare cases will CaF_2 be used in the blast furnace. While it makes a slag fluid, it is very expensive and has a strongly corrosive action on the furnace lining. If pig iron running high in P is desired, as, for example, in the basic Bessemer process, it may be necessary to add apatite, $3Ca_3P_2O_8 + Ca(Cl.F)_2$, with 41.0 and 42.3 per cent. P_2O_5, or other phosphates.

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[1] "Transactions Institution of Mining Engineers," 1903, XXVI, p. 117.

[2] Elbers, "Engineering and Mining Journal," April 18, May 9, 1896.

[3] Elbers, "Engineering and Mining Journal," January 19, 26, 1895; Firmstone, "Transactions American Institute of Mining Engineers," XXIV, p. 498; Foster, *op. cit.*, XXIX, p. 562.

ANALYSES OF LIMESTONES.

	Durham furnace, Riegelsville, Pennsylvania.[1]	Colorado Fuel and Iron Company.[2]	Eastern Pennsylvania.[3]		Dover, Ohio.[4]	Alabama.[5]		Tennessee.[6]	Pennsylvania.[7]	Ohio.[8]	Logansport, Indiana.[9]	Illinois.[9]	Nova Scotia.[9]
Fe	0.54	1.23	. . .	1.27	. . .	0.42	1.40	0.69	. . .	. . .	. . .	. . .	0.41
Al_2O_3	2.00	1.71	. . .		2.42	0.40			. . .	. . .	.72	.78	0.12
CaO	27.62	49.06	44.52	30.42	50.30	30.31	54.15	42.49	. . .	. . .	49.28	48.58	52.58
MgO	18.75	0.56	5.45	20.95	0.46	20.71	. . .	8.60	. . .	. . .	2.89	2.19	1.16
SiO_2	8.84	7.68	7.18	2.07	5.44	1.50	2.50	4.60	. . .	. . .	4.61	7.36	2.10
P	0.027	0.033	. . .	. . .	. . .	. . .	. . .	0.022	. . .	. . .	. . .	. . .	0.01
S	0.020	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	0.03
Mn	. . .	0.32	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	0.35
H_2O	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	0.18

[1] "Transactions American Institute of Mining Engineers," XXI, p. 348.
[2] *Op. cit.*, XXIII, p. 580.
[3] *Op. cit.*, XXIV, p. 499.
[4] *Op. cit.*, XXVII, p. 480.
[5] Phillips, "Iron Making," 2d ed., p. 321.
[6] "Transactions American Institute of Mining Engineers," XV, pp. 2, 13.
[7] "Second Geological Survey," MM, p. 281.
[8] "Geological Survey of Ohio," VI, pp. 716–769.
[9] Private Notes.

SECTION 7. **Fuels.** The fuels used in the blast furnace are coke, anthracite, charcoal, and in a few instances non-caking bituminous coal high in volatile matter.

(*a*) *Coke.*[1] Coke is the leading blast-furnace fuel. It is the strongest of the blast-furnace fuels. Its properties vary with the nature of the coal and the method of manufacture. Its color is silver gray to dull black. The surface is rough. It is more or less porous. Connelsville coke[2] made in the beehive showed 44.93 per cent. cell-walls and 56.07 per cent. cell-spaces, while Otto-Hoffmann coke from the sides and the bottom of retort gave 61.13 per cent. and 77.22 per cent. cell-walls, and 38.81 per cent. and 22.78 per cent. cell-spaces. Coke is hygroscopic (beehive coke contains 2–3 per cent., retort coke, 5–6 per cent. H_2O), hard (beehive, 3.00–3.60 Mohs scale), strong. Fronheiser[3] obtained the following data by exposing in a tube anthracite and different kinds of coke to hot carbonic acid gas. They show that the weaker the coke the more readily is it oxidized by CO_2.

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[1] O. Simmersbach and W. C. Anderson, "The Chemistry of Coke." Edinburgh, 1899. J. D. Weeks, "Report on the Manufacture of Coke," from Tenth Census, United States, 1880. Fulton, "Coke." Scranton, Pennsylvania, 1895.

[2] Fulton, "Coke," p. 230.

[3] Fulton, *op. cit.*, p. 251.

OXIDATION OF FUEL BY HOT CARBONIC ACID.

	CO_2	CO	Hardness.	Ultimate compressive strength per cubic inch without crushing.
Anthracite	96.0	4.0	2.5	3,000
Connelsville coke, Otto-Hoffmann .	94.5	5.5	3.7	2,260
Connelsville coke, beehive . . .	91.9	9.0	3.0	1,204
Morris Run, Semet-Solvay . . .	88.8	11.2	2.6	1,360
Bennington, beehive	86.1	13.9	2.4	848

The ULTIMATE ANALYSIS of coke shows the following composition: —

Country.	C	H	O	N	Ash.	S
Germany, Ruhr[1]	83.487	0.737	5.467	. . .	10.309	
Germany, Ruhr[1]	91.772	1.225	0.040	. . .	6.933	
England, Dukinfield[2]	85.480	0.520	. . .	. . .	9.150	
England, Durham (Cardiff)[3]	91.580	0.230	1.310	. . .	6.860	
Belgium, Mons[3]	80.850	0.510	2.130	. . .	16.510	
Belgium, Séraing[3]	91.300	0.330	2.17		6.200	. . .

[1] Muck, "Steinkohlenchemie," p. 203.
[2] Percy, "Fuel," p. 417.
[3] Weeks, "Coke," p. 72.

In this country only the PROXIMATE ANALYSIS is made:—

Locality.	GRAMS IN 1 CUBIC INCH.		POUNDS IN 1 CUBIC FOOT.		PERCENTAGE BY VOLUME.		Compressive strength per cubic inch, ¼ ultimate strength.	Height of furnace charge supported without breaking.	Order in cell space.
	Dry.	Wet.	Dry.	Wet.	Coke.	Cells.			
Connelsville, Pennsylvania .	12.51	21.62	47.69	82.20	43.90	56.07	272	108	1
Connelsville, Pennsylvania .	14.99	23.73	57.13	89.45	48.24	51.76	387	155	1
Connelsville, Pennsylvania .	17.57	22.62	66.93	86.18	69.17	30.83	702	281	1¼
Pocahontas, Virginia . . .	15.67	23.53	59.68	89.64	52.07	47.93	236	94	1
Tuscarawas, Ohio	11.14	20.32	42.43	61.61	38.81	61.19	156	62	1
Caledonia, Pennsylvania .	12.10	21.80	46.12	83.07	40.83	59.17	149	59	1
Caledonia, Pennsylvania .	13.57	22.41	51.69	85.38	46.07	53.93	209	84	1
Blocton, Alabama . . .	14.32	22.52	54.56	85.80	49.97	50.03	409	164	1.2
Pineville, Kentucky . . .	14.10	22.42	53.73	84.73	50.37	49.63	227	91	1.2
Big Stone Gap, Virginia .	11.89	21.18	45.31	80.70	43.34	56.66	245	98	1¼
Big Stone Gap, Virginia .	12.20	21.02	46.49	80.09	46.22	53.78	326	131	1¼
Cardiff, Wales	12.90	22.27	49.16	84.86	42.76	57.24	231	92	1

Locality.	Hardness.	Specific gravity.	PROXIMATE ANALYSIS.						Remarks.
			F. C.	H_2O	Ash.	S	P	V. M.	
Connelsville, Pennsylvania .	3.0	1.74	86.88	0.79	11.54	0.695	.005	1.31	Beehive.
Connelsville, Pennsylvania .	3.1	1.89	86.38	0.03	13.08	0.68	.015	0.51	Hüssner.
Connelsville, Pennsylvania .	3.3	1.90	85.60	0.12	12.26	0.52	.006	2.02	Otto-Hoffmann.
Pocahontas, Virginia . . .	2.5	1.83	92.69	0.34	5.88	0.74	.006	0.34	Beehive.
Tuscarawas, Ohio	2.5	1.75	84.21	0.13	12.91	0.37	.015	2.75	Hüssner.
Caledonia, Pennsylvania .	3.0	1.81	87.89	0.13	9.42	1.57	.024	0.97	Beehive, 48 hours.
Caledonia, Pennsylvania .	3.0	1.80	. . .	. . .	. . .	. . .	. . .	. . .	Beehive, 72 hours.
Blocton, Alabama . . .	2.6	1.75	92.76	0.153	6.94	0.74	.006	0.81	Beehive.
Pineville, Kentucky . . .	2.6	1.71	94.66	1.14	3.78	0.59	.007	0.41	Beehive.
Big Stone Gap, Virginia .	2.5	1.67	92.05	0.29	5.60	0.74	.009	1.32	Beehive, 48 hours.
Big Stone Gap, Virginia .	2.7	1.61	93.81	0.63	3.63	1.01	.005	1.93	Beehive, 72 hours.
Cardiff, Wales	2.5	1.84	95.00	0.06	4.26	0.68	.018	. . .	Beehive.

The percentage of S in coke is liable to be high.[1] The calorific power of coke is about 8,000 calories. It was believed that beehive-coke was better suited for a blast-furnace fuel than retort-oven coke, but parallel use[2] has disproved this. In fact, retort ovens will gradually supersede the crude beehives, which give a lower yield in coke and allow the by-products to go to waste.

(*b*) ***Anthracite*** is used only in eastern Pennsylvania. It is hard, compact, brittle, jet black, has a conchoidal fracture, and a luster from vitreous to submetallic. It is dense, and hence difficult to ignite; therefore anthracite furnaces cannot be driven as fast as coke furnaces. It further decrepitates in the blast furnace, and gives trouble from infusible mixtures of fine coal and ash. These troubles may be so great as to obstruct the passage of the blast when, *e.g.*, the tuyères have had to be removed and the dirt raked out with other stock so as to secure a passage for blast. The size should not exceed a 4-inch cube. The fuel consumption is greater than with coke. The calorific power varies from 9,000 to 9,500 calories.

ANALYSES OF ANTHRACITE.[3]

FIELD.	EASTERN MIDDLE.		WESTERN MIDDLE.				SOUTHERN.		NORTHERN OR WYOMING.
District.	Hazelton.		Mahanoy, Shamokin.				Lehigh, Pottsville.		Wilkesbarre, Scranton.
Bed.	Wharton.	Mammoth.	Primrose.	Mammoth.	Buck Mt.	Seam Ft.	Primrose F. bed.	Mammoth.	Mammoth.
H_2O	3.713	4.119	3.541	3.163	3.042	3.410	3.008	3.087	3.421
V. M.	3.080	3.084	3.716	3.717	3.949	3.978	4.125	4.275	4.381
F. C.	80.404	86.379	81.590	81.143	82.662	80.868	87.982	83.813	83.268
S	0.585	0.496	0.499	0.899	0.462	0.512	0.506	0.041	0.727
Ash	6.218	5.922	10.654	11.078	9.885	11.232	4.379	8.184	8.203
				100.00					
Specific gravity .	1.620	1.617	1.654	1.657	1.607	1.651	1.584	1.631	1.575

(*c*) *Charcoal* is used in this country as a blast-furnace fuel principally in Michigan and Wisconsin; in Europe Sweden uses almost exclusively charcoal. It is the most porous and the purest fuel; but it is weak, and breaks up considerably in the blast furnace. Its physical properties depend on the character of the wood and the mode of charring. Good charcoal is black, lustrous, hard, sonorous; has a conchoidal fracture; soils the fingers only a little, and burns in small pieces without flame or smoke. It is very porous, consisting of a large number of small, elongated cells joined to each other by porous walls.

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[1] Simmersbach, "Stahl und Eisen," 1903, p. 163.

[2] Fulton, "Coke," p. 338.

[3] Ashburner, "Mineral Resources of the United States," 1883–84, p. 69; "Transactions American Institute of Mining Engineers," XIV, p. 717.

Analyses of anthracites used as blast-furnace fuel in Southern Russia: "Stahl und Eisen," 1901, p. 1,091.

ULTIMATE ANALYSES OF CHARCOAL.

Charcoal.	C	O	H	Total.	Ash.	H_2O
Air dry	75.5	12.0	2.5	90.0	1.0	9.0
Absolutely dry . .	83.0	13.2	2.7	98.9	1.0	. . .

The ash consists mainly of alkali and alkaline earth carbonates. The calorific power is about 6,900 calories. The size should be over 1 inch cube.

(*d*) *Bituminous Coal.* Non-caking coals rich in oxygen form the blast-furnace fuels of parts of Scotland and England. Formerly they were used in Ohio. The English coals used have 55 per cent. fixed carbon and a calorific power of 6,100 calories.[1] Ohio block coals,[2] from Hocking Valley, have 53–60 per cent. fixed carbon; those from Mahoning Valley 63–66 per cent. fixed carbon. Bituminous coals give off their volatile matter in the upper parts of the blast furnace, evolving large volumes of gas. The heat consumed by this destructive distillation of coal has to be made up by an extra amount of fuel. In Scotland the gases are passed through condensers and scrubbers,[3] thereby tar and ammonia are recovered as a valuable by-product.

Taking the three leading fuels, they stand as to porosity and purity in the order: charcoal, coke, anthracite; as to strength the order is coke, anthracite, charcoal.

The selection and valuation of iron ores, limestones, and fuels for blast furnaces is discussed by Gordon,[4] Gaines,[5] and List.[6]

SECTION 8. **The Blast-furnace Plant.**[7] The location of the blast-furnace plant is usually

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[1] Turner, "Metallurgy of Iron," p. 159.

[2] "Transactions American Institute of Mining Engineers," II, p. 275.

[3] "Stahl und Eisen," 1902, p. 545; Sexton, "Metallurgy of Iron and Steel," 1902, p. 156.

[4] "Transactions American Institute of Mining Engineers," XXI, p. 61.

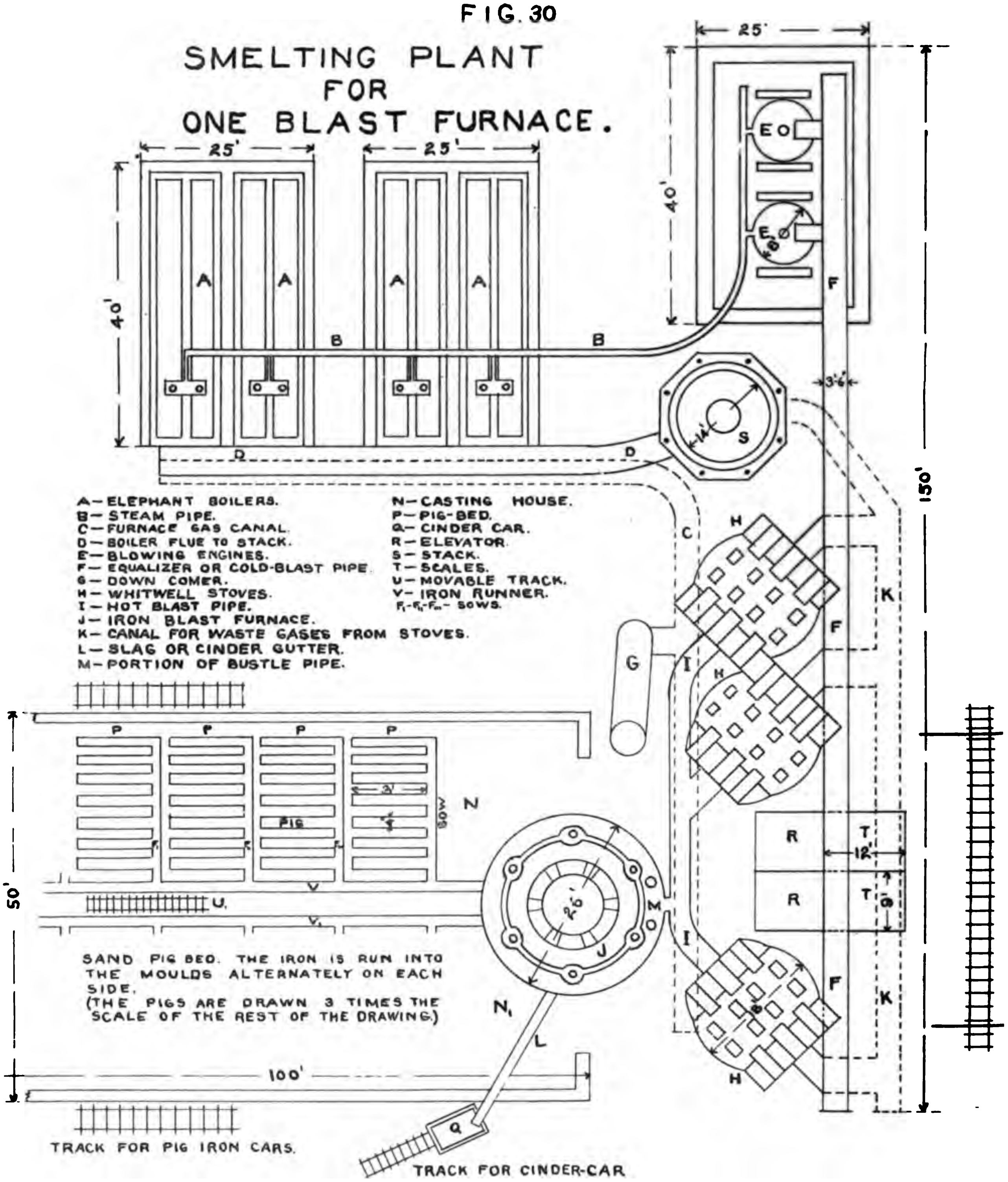
[5] "Iron Age," April 14, 1904.

[6] "Stahl und Eisen," 1901, p. 1343.

[7] *Blast-furnace Plants.* **First.**—*In General.* White, "Iron Age," September 11, 18, November 6, 1890. Pilkington, "Iron Age," December 3, 1891. Stevenson, "Engineer," XCIV, p. 248. **Second.**—*Handling Ore.* Wedding, "Stahl und Eisen," 1891, p. 459. Robinson, "Cassier's Magazine," 1902, XXII, p. 376. Fawcett, "Engineering and Mining Journal," 1900, LXIX, pp. 77, 107. Sahlin, "Transactions American Institute of Mining Engineers," XXVII, p. 3. Frahm, "Stahl und Eisen," 1900, pp. 513, 597, 653, 698, 750, 798. Johnston, "Stahl und Eisen," 1901, p. 14. Langheinrich, "Stahl und Eisen," 1901, p. 14. Bessemer Road, "Iron Trade Review," March 31, 1898. Hulett Machine, "Iron Age," April 5, 1900, and "Iron Trade Review," November 13, 1902. Hoover and Mason Machine, "Iron Age," September 4, 11, 1902; "Iron Trade Review," October 15, 1903. W. Kennedy Skip Hoist, "Iron Trade Review," 1901, XXXIV, p. 11. E. G. Rust Skip Hoist, "Iron Age," 1900, LXVI, p. 10. Brown Hoist, "Iron Age," 1898, May 26. **Third.**—*Charcoal Blast Furnaces.* Grand River, Kentucky, "Iron Age," August 25, 1897. Hinkle Furnace, Ashland, Wisconsin, "Iron Age," March 28, 1889. Gladstone Furnace, Michigan, "Journal Iron and Steel Institute," 1896, I, p. 409. **Fourth.**—*Coke* (*Anthracite*) *Blast Furnaces.* South Chicago, "Stahl und Eisen," 1887, p. 698; "Journal Iron and Steel Institute," 1887, I, p. 163, Plate XVI. Ensley, Alabama, "Stahl und Eisen," 1888, p. 6; "Iron Age," 1887, November 18. Monongahela, two furnaces, "Iron Age," August 7, 1890. Buffalo Furnace Company, "Iron Age," April 13, 1893. Duquesne Furnace, "Iron Age," March 25, 1897. Lorain Furnaces, "Engineering and Mining Journal," August 13, 1898, May 4, 1901; "Iron Trade Review," July 7, 1898, October 19, 1899. Ohio Works of the National Steel Company, Youngstown, Ohio, "Iron Age," and "Iron Trade Review," December 21, 1899. Colorado Fuel and Iron Company, Pueblo, Colorado, "Iron Age," August 14, p. 102. Sharon Steel Company, Sharon, Pennsylvania, "Iron Age," July 4, 1901. Toledo Furnace Company, "Iron Trade Review," November 5, 1903. Eliza Furnaces, Pittsburg, "Iron Age," October 31, 1901. Iroquois Furnaces, South Chicago, "Iron Age," November 28, 1901; "Iron Trade Review," November 28, 1901. St. Clair Furnaces, Clairton, Pennsylvania, "Iron Age," March 20, 1902. Buffalo and Susquehanna Iron Company, "Iron Age," August 6, 1903. La Belle Iron Works, Steubenville, Ohio, "Iron Age," October 8, 1903. International Harvester Company, South Chicago, "Iron Trade Review," October 15, 1903. Clairton Steel Company, Clairton, Pennsylvania, "Iron Trade Review," November 19, 1903. Lackawanna Steel Company, Buffalo, "Engineering News," 1903; "Iron Trade Review," December 31, 1903; "Iron Age," January 7, 1904. Adrian Furnaces of the Rochester and Pittsburg Coal and Iron Company, "Iron Trade Review," May 5, 1904.

FIG. 30

SMELTING PLANT
FOR
ONE BLAST FURNACE.

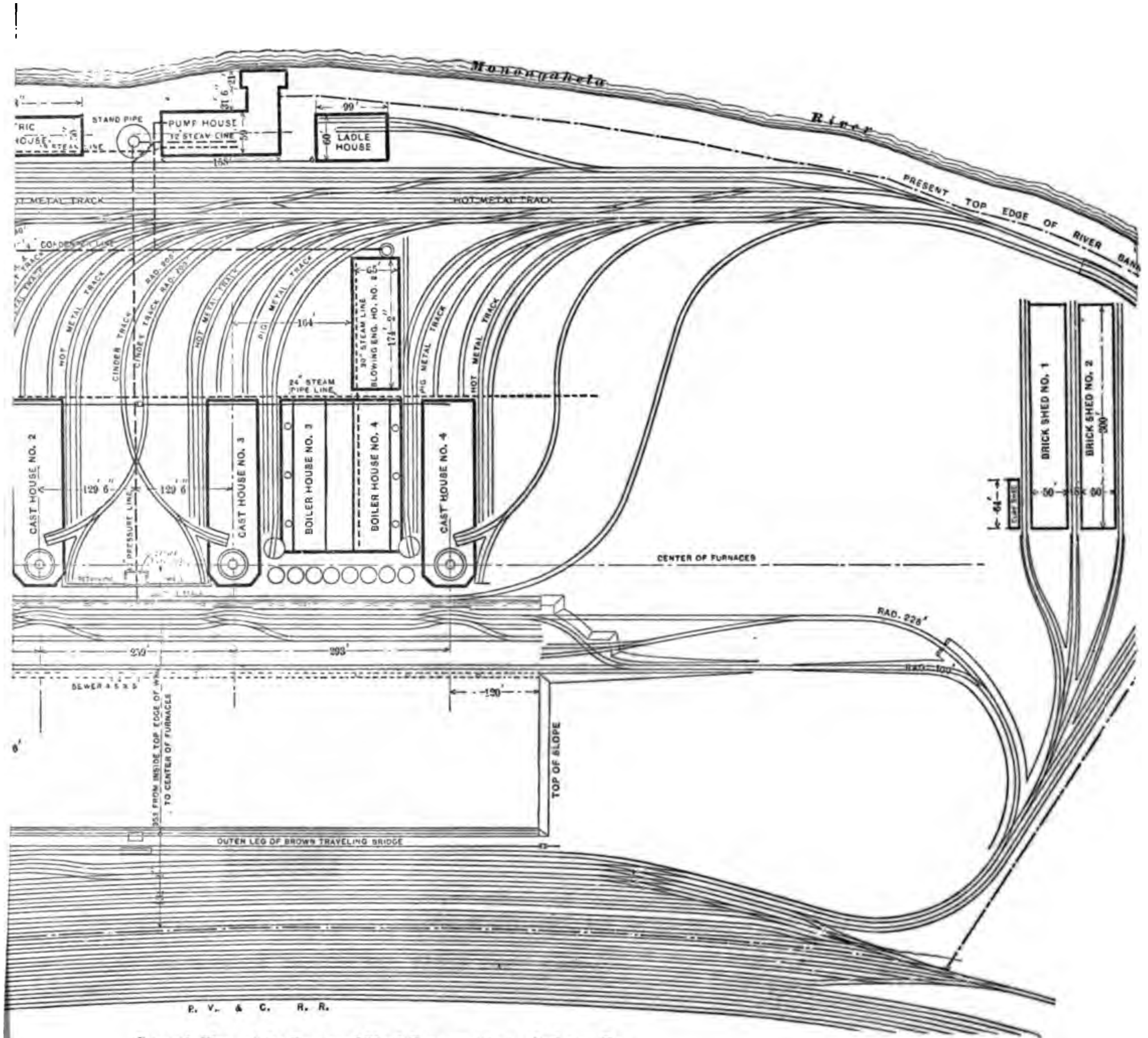


determined by the cost of assembling the raw materials (ore, flux, fuels, and refractories) and of getting the product to market. A modern large iron blast furnace produces in twenty-four hours 500 tons Bessemer pig iron and about 300 tons slag. For this it requires 833 tons 60-per cent. iron ore, 475–500 tons coke, and about 200 tons limestone. The site should furnish enough level space for buildings, yards, etc., a ready means for the disposal of the slag, a good water supply, and solid grounds for foundations. In view of the small capacity and the limited facilities for hauling and hoisting, blast furnaces were formerly erected on a hillside, which permitted wheeling the charge to the top of the furnace. A modern furnace stands on a level plane and the materials are hoisted to the throat by elevators. The level site requires little excavating and no retaining walls. It is compact; necessitates little tramming and handling by manual labor — in one word is cheaper than a sloping site.

(1) *General Plan of a Smelting Plant with One Blast Furnace* (Figure 30, Witherow and Gordon). Ore, flux, and fuel arrive in the stock house on cars running on a trestle, and are emptied into bins below. From these they are discharged or shoveled into trucks and go to the scales, *T*, the elevator, *R*, and the blast furnace, *J*. This produces pig iron, slag, and gases. The pig iron is tapped at intervals, flowing alternately into one of the two pig beds, *NN'*, of the casting house, *K*. It flows down one of the iron runners, *VV'*, passes into the lowest feeder (or sow), *F*, thence into the sand or iron molds, *P'*, forming semi-cylindrical bars (pigs). When the lowest row of molds has been filled, the lower end of the runner is shut off by a cast-iron gate (similar to a spade) and the metal conducted into the next higher sow, *F*, and so on. When solidified and superficially cooled the pigs and the iron in the sows and runners are broken with sledges and carried onto trucks running on a movable track, *U*, and thence loaded into pig-iron cars. The slag is tapped at intervals, conducted by the slag or cinder gutter (cinder runner), *L*, into the cinder car, *Q*, and hauled away. The gases leave the furnace through the downcomer, *G*; part of them are burned in the stoves, *H*, and part under the boilers, *A*. The burnt gases from the stoves pass off through small chimneys on the tops (with most stoves they descend and travel through an underground main to a central stack). The gases from boilers enter the stack, *S*. Steam from the boilers drives the blowing engines, *E*; and the compressed air travels through pipe *F* to the heating stoves, *H*, and thence through the bustle pipe *P* to the blast furnace. Pumps are not shown in the plan.

(2) *The Duquesne Blast-furnace Plant, Cochran, Pennsylvania*[1] (Figures 31–33). This plant, shown in plan and elevation in Figures 30 and 31, has four blast furnaces 97 feet high with a 14-foot crucible, a 22-foot bosh, and a 12-foot 6-inch throat, producing each 500–600 tons Bessemer pig in twenty-four hours, which requires about 2,000 tons materials. The stock yard, 1,085 x 300 feet, extends the whole length of the furnace plant. Its effective width is 226 feet. It is excavated 26 feet below the main level, and has a capacity of 600,000 tons ore. This large capacity is necessary, as the plant is run mainly on Lake Superior ores, which can be shipped only part of the year. The stock yard is spanned by three Brown hoisting and conveying cranes, each handling 1,500–2,000 tons ore in ten hours. There are two series of bins — thirty-six double ore bins on the stock yard side, and twenty-four single coke and limestone bins on the furnace side. The ore arrives in cars with drop bottoms. They are run over the track on the yard (delivery) side if the ore is to be stocked; on the furnace (consumption) side if it is

[1] "Iron Age," March 25, 1897.



General Plan of the Duquesne Blast Furnace Plant, Cochran, Pa.

Fig. 31.

to be used directly in the blast furnace. That dropped into the bins on the yard side is withdrawn from these into buckets, which are picked up by the Brown conveyor and emptied into the ore pile. The ore dropped into the bins on the furnace side is withdrawn from these, when needed, into buckets (10,000 pounds capacity) resting upon cars provided with weighing scales. A train of cars when filled is switched to the foot of the furnace hoist, which picks up a bucket, conveys it to the top of the furnace, and there dumps its contents. The coke and limestone bins, which receive also mill cinder or special kinds of ore, face the ore bins on the furnace side. There are for each furnace one pair of bins for coke, one pair for limestone, and one for mill cinder and special ores. These materials are delivered from a track running over the bins as needed, and no attempt is made to keep any stock on hand. The total capacity is coke, 3,600 tons; limestone, 2,200 tons; various ores, 9,500 tons. Coke and limestone are discharged into cylindrical buckets similar to the ore buckets. They hold about 4,000 pounds coke.

The four blast furnaces (see blast-furnace table) produce together 2,000 tons pig iron (3 Bessemer, 1 basic pig) in twenty-four hours. On week days all the metal is run into metal ladles standing on the hot-metal track. The Bessemer metal goes to the mixer, whence it is delivered to the converters. The basic metal goes in part to a mixer, in part to a casting machine, or it is tapped into the cast-iron molds in the cast house, which is 219 x 70 feet. At the lower end of each cast house is a pig-breaking machine. The pigs are taken from the beds by two 10-ton overhead electric cranes of 32-foot span and are brought to the machine, broken, and discharged into cars standing on the pig-metal track. A 5-ton overhead electric crane runs along the centre of the cast house; it serves to convey scrap, etc., to the hoist house. The slag is run into ladles on cars running on the cinder track.

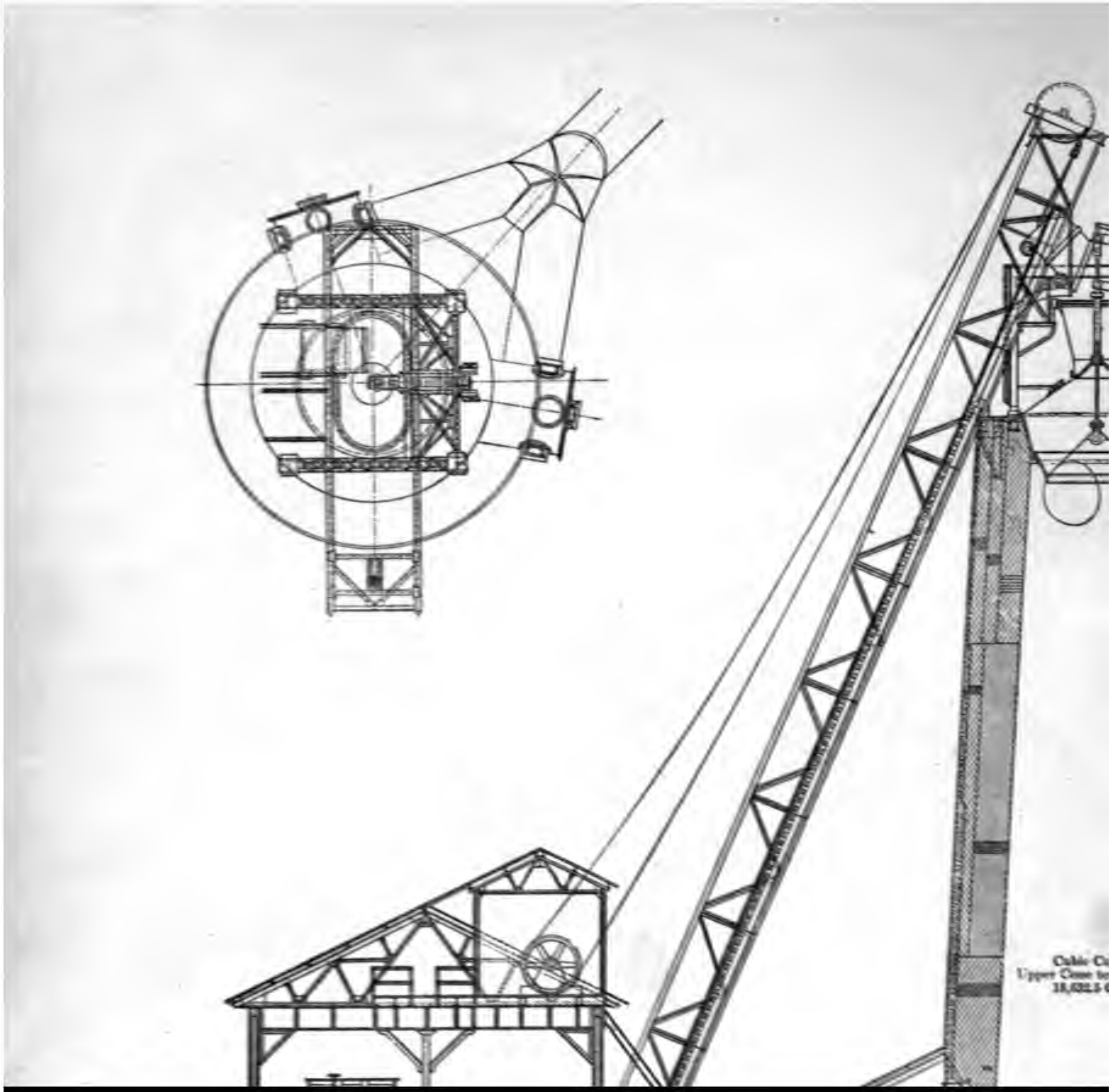
The gases, taken off by six flues, are collected in two inclined (45°) pipes, each of which has a 30-inch explosion door and a 30-inch bleeder. The pipes pass into the downcomer, which is inclined to the same angle and ends in the dust catcher, 28 feet in diameter and 40 feet in height. It has explosion doors. On the top of the dust catcher is a bleeder stack to assist in decreasing the velocity of the gases.

Stoves. Each furnace has four Kennedy-Cowper stoves (21 feet in diameter and 97 feet in height).

The Boilers are of the Babcock and Wilcox type. They are arranged in four batteries of 3,000 horse power, forming two groups of 6,000 horse power, the individual boiler being of 250 horse power. They are gas-fired, the gas entering at the bottom of a brick-lined combustion chamber (dog house) projecting 3 feet from the face of the boiler.

Blowing Engines. There are for each pair of furnaces five E. P. Allis and Company vertical cross-compound condensing beam-type engines with a 40-inch high pressure and a 78-inch low pressure steam cylinder, two 76-inch air cylinders, and a 60-inch stroke. The air-inlet valves are positive, the outlet valves automatic. One of the five engines is kept as a reserve for contingencies. The engines deliver 610 cubic feet air per revolution; their usual speed is twenty-eight revolutions per minute, which gives a blast pressure of 15 pounds at the tuyères.

The Pumping Plant consists of four vertical compound condensing pumps with 22-inch high pressure and 44-inch low pressure steam cylinders, a 14-inch pump cylinder, and 36-inch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stroke. The water is forced into a standpipe through a 42-inch pipe. The capacity of the plant is 20 mill gallons per day.

The Electric Plant contains three 500 horse power generators, operated in parallel, supplying power to a large number of direct-current 220-volt motors, to six direct constant-current 60-light arc dynamos, and to a 45-kilowatts alternating single-phase direct-connected generator, supplying current to about nine hundred incandescent lamps.

The Eliza Blast Furnaces, Pittsburg, Pennsylvania (Figures 34 and 35).[1] This plant has four blast furnaces 100 feet high, 22-foot bosh, 14-foot 6-inch crucible, and 13-foot throat, each of which produces about 500 tons pig iron in twenty-four hours.

The Ore Yard has an area of 750 x 300 feet and holds 550,000 tons iron ore. It has nine elevated tracks, 700 feet long, resting on columns 40 feet high. The ore arrives on these tracks in bottom-discharge cars, is dumped and stored until needed. From the yard it is loaded into cars from the piles by means of locomotive steam shovels, which run on tracks laid on the ground floor at right angles to elevated tracks and 25 feet apart from one another. The cars supply the needs of the ore bins in the stock house near the furnaces. Coke and limestone are brought direct to the bins.

The Stock House, 1,110 x 40 feet, has two rows of bins, 128 in all, with sloping (45°) bottoms (maple backed by oak) and vertical fronts (½-inch steel plate), which face one another. The framework is made of massive structural steel. One row of bins (the higher one) serves for coke and limestone, the other for ore. The contents are discharged through sliding doors, worked with compressed air, into electrically driven cars, two to a furnace, running on a ground track which extends the entire length of the stock house. Ore, flux, and fuel are weighed while they fill the larries, and are then discharged into one of the two skip buckets, 5 feet square, of a furnace, raised to the throat on an incline, 60° with the horizontal, and automatically dumped into the hopper of the furnace. Two of the skips are drawn by 150-horse power electric motors and two by Otis-Crane's hoisting engines, with 14 x 16-inch cylinders.

The upper hopper of the furnace is oblong and closed by a small upper bell. It rests upon the gas-sealing hood, which is supported by the large cylindrical hopper of the lower bell. Both bells are worked independently from the hoisting-engine house. The leading dimensions of the furnace are given in the drawing. There are ten columns to support the shaft, twenty tuyères 6 inches in diameter, ten rows of bronze bosh-cooling plates with cast-iron boxes, a cast-iron water-cooled hearth jacket, two elliptical gas outlets at the throat 116° apart, and a pipe encircling the rest of the furnace which has six explosion doors 34 inches in diameter. The outlets connect with two downcomers 73 inches in diameter, inclined 35° from the vertical, which are joined 40 feet farther down into a single pipe 98 inches in diameter, ending in a dust catcher 21 feet in diameter. The gas flues leading to the stoves and boilers have dust pockets 3 feet in diameter, closed by bell and hopper.

A blast furnace has four Cowper-Kennedy stoves 22 feet in diameter and 108 feet high; one stack 12 feet inside diameter and 220 feet high furnishes the draught for eight stoves.

The Plant is a direct-metal plant. A furnace is tapped every four hours and the metal collected in three (?) hot-metal ladles, each holding 20 tons. A ladle can handle 1,000 tons molten metal in twenty-four hours.[2]

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[1] "Iron Age," October 31, 1901.

[2] Drawings of ladle and car in original.

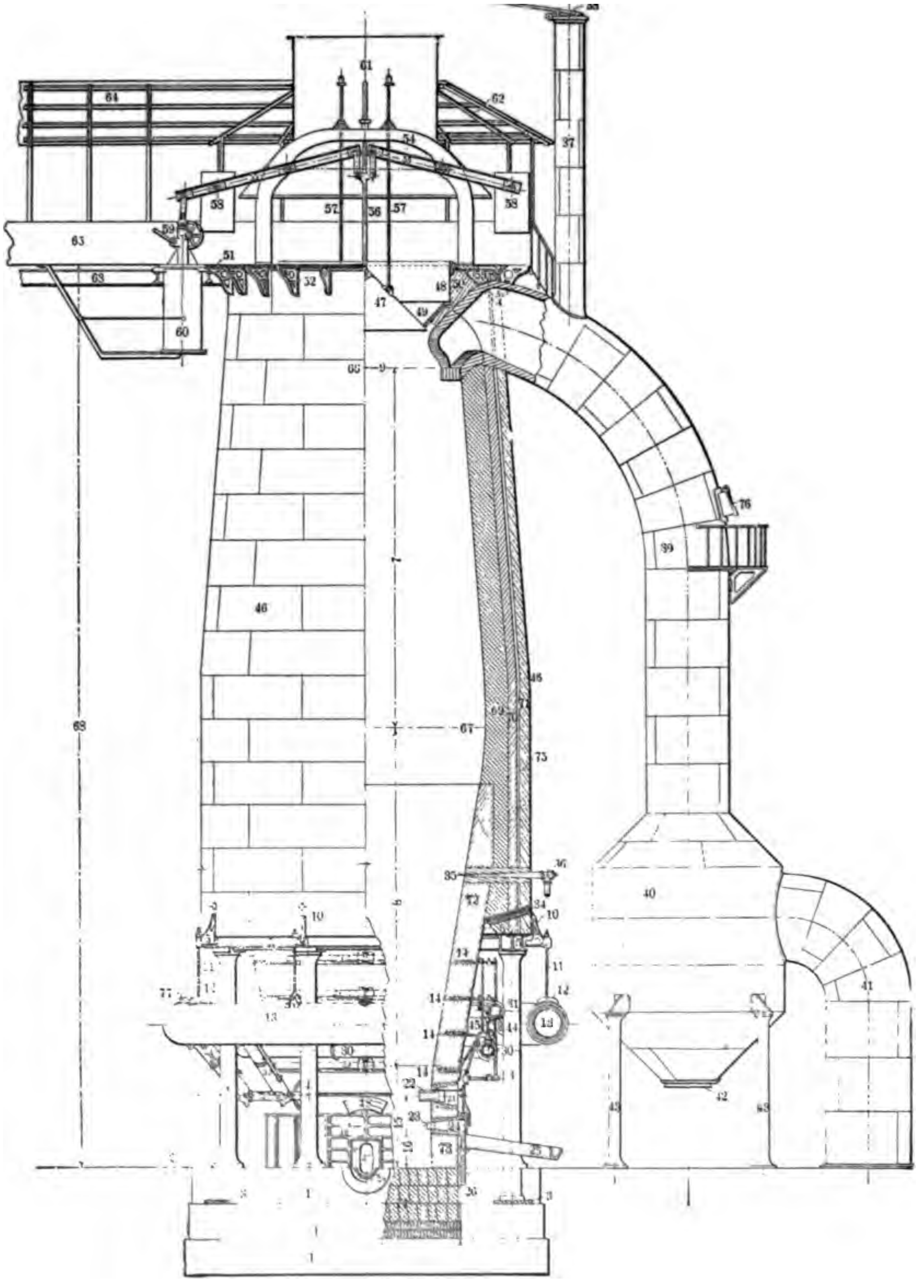

Fig. 30.

*1 Foundation.
*2 Hearth-level
3 Base-plate.
*4 Columns.
5 Mantel beams.
6 Mantel-plate.
*7 Shaft
*8 Boshes.
9 Throat.
10 Bustle-pipe bracket.
11 Bustle-pipe hanger
12 Bustle-pipe saddle.
*13 Bustle-pipe.
*14 Bosh-plates.
*15 Hearth
*16 Crucible
17 Tymp.
18 Front or Tymp-arch.
19 Dam.
20 False or monkey dam.
*21 Tuyère.
*22 Nozzle.
*23 Slag or Cinder notch
*24 Slag or Cinder-notch arch
*25 Slag or Cinder-notch runner.
*26 Hearth-jacket.
*27 Tap-hole.
*28 Iron runner.
29 Crinoline bands.
30 Waste water pipe.
31 Feed-water pipe.
*32 Belly pipe.
*33 Tuyère stock.
34 Ventilating-ports.
35 Bosh ports.
*36 Bosh tuyère.
*37 Bleeder.
*38 Bleeder-valve.
*39 Down-comer.
*40 Dust-catcher.
*41 Dust catcher goose-neck
*42 Dust-catcher bell.
43 Dust-catcher columns.
44 Feed-water-pipe bracket.
45 Waste-water-pipe hanger.
*46 Furnace-shell.
*47 Bell.
*48 Hopper.
49 Hopper extension-ring.
50 Hopper lip-ring.
*51 Top platform.
52 Platform-brackets.
53 Hopper-brackets.
*54 Bell-arch.
*55 Bell-beams.
56 Bell lifting-rod.
57 Bell safety-rods.
58 Bell counter-weights.
59 Bell-winch.
60 Bell air-cylinder.
61 Waste-gas flume.
62 Furnace-canopy.
63 Bridge-beams.
64 Bridge-roof.
65 Platform-guards.
*66 Throat- or stock-lin
*67 Bosh.
*68 Height.
*69 In-walls.
*70 In-walls backing.
*71 In-walls filling.
*72 Bosh-walls.
*73 Crucible-walls.
*74 Hearth-blocks.
75 Expansion-space.
76 Explosion-door.
77 Bustle-pipe man-hol

Figures printed with asterisk represent the leading parts.

The cinder is run from the furnace into a pit 20 feet long by 30 feet wide by 24 feet deep, filled 10 feet with water, together with a flat stream of water. The granulated slag is removed with a clam-shell bucket[1] of 1½ cubic yards capacity onto cars.

The Power Plant has fifty-six Laughlin boilers, rated each at 350 horse power.

The Blast is furnished by the following engines: one pair Allis horizontal engines 42 x 84 x 72 inches, displacement per revolution 923.6 cubic feet; one Southwark engine 42 x 84 x 60 inches, displacement 348.6 cubic feet; four Allis vertical engines 40 x 79 x 60 inches, displacement 314 cubic feet; one Allis vertical engine 42 x 79 x 60 inches, displacement 314 cubic feet; seven pairs of Allis cross-compound engines, 42 x 80 inches steam, 87 x 87 inches air cylinders, stroke 60 inches, displacement 772.8 cubic feet.

The Pump House contains two 6,000,000- and 12,000,000-gallon triple-expansion, high-duty pumping engines.

The Electric Plant consists of three Westinghouse, 200-kilowatt direct-current, 220-volt generators.

Shops, Feed-water Heaters, and Condensers can be mentioned only by name.

SECTION 9. **The Blast Furnace in General** (Figure 36). (1) *The Shape.* The blast furnace is a brick shaft, usually circular in horizontal section. It is charged at the top with alternate layers of fuel and ore and flux. Near the bottom the blast is introduced through tuyères (21). It burns the fuel; the heat generated causes the charge to melt at this level where the pig iron (specific gravity 7.3) and the slag or cinder (specific gravity 2.8) separate to be tapped through the metal tap (27) and the cinder notch (23). The current of hot gases ascending the shaft gives up its heat to the descending ore charge, and at the same time exerts a reducing action upon it. The gases leave the furnace at the throat (9), descend in the down-comer (39), pass through the dust catcher (40) into the goose neck (41), and thence to the gas main leading to the boilers (gas engines) and heating stoves, where the CO they contain furnishes the fuel required for heating.

The furnace consists of three parts: the upper part, the trunk or shaft (7) having the form of a truncated cone, rests upon columns; the lower part, the cylindrical hearth (15), rests directly upon the foundations; and the middle part, the bosh (8), having the form of an inverted truncated cone, rests upon the hearth walls. The smallest section is that of the hearth, as, on account of the coke being burnt and the deoxidized ore and flux being melted, the original volume of the charge has been greatly reduced.[2] The object of having a bosh is to retard the quickly ascending gas current, and to allow it to act upon the descending charge. If the inverted cone form were carried to the throat it would interfere with the regular descent of the charges, because, the friction near the walls being greater than in the centre, the central part would descend more quickly than that near the sides. In order to decrease the friction at the walls and to permit the closing of the throat by a bell, the shaft is narrowed from the bosh line (67) to the throat. Cylindrical furnaces have proved a failure.[3]

(2) *Capacity or Smelting Power.*[4] The smelting power of a furnace depends, other things

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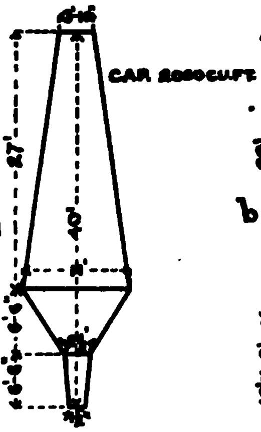
[1] "Iron Age," October 23, 1902.

[2] Furnace lines: "Stahl und Eisen," 1901, p. 1090; "Bulletin de L'industrie Minérale," 1902, I, p. 493; "Oesterreichische Zeitschrift für Berg- und Hütten-wesen," 1903, pp. 51, 188; see also blast-furnace table.

[3] Taylor, "Transactions American Institute of Mining Engineers," XIII, p. 489.

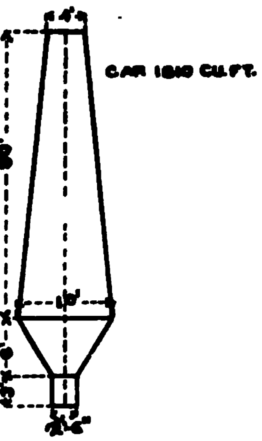
[4] Birkinbine, "Iron Age," January 2, 1896.

a



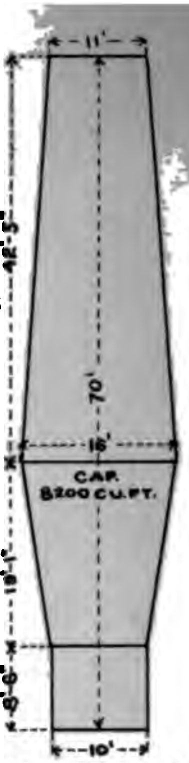
CHICKIES, NEAR
COLUMBIA, PA.
ANTHRACITE.

b



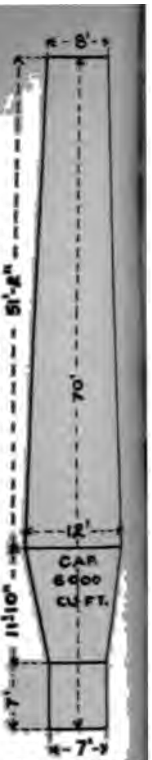
KY. AND PA.
CHARCOAL.

c



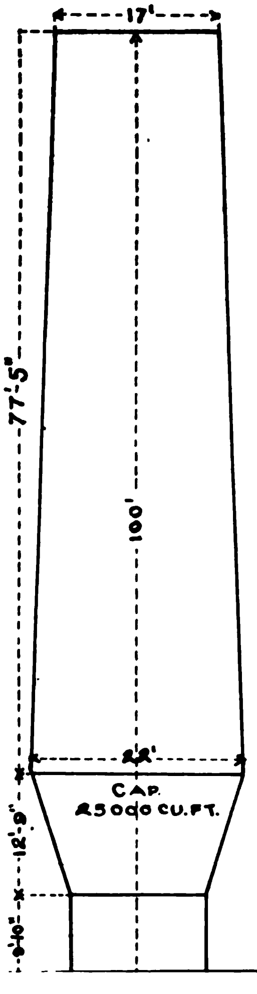
WARWICK IRON AND STEEL CO.
POTTSTOWN, PA.
ANTHRACITE.

d



GLADSTONE
MICH.
CHARCOAL.

e



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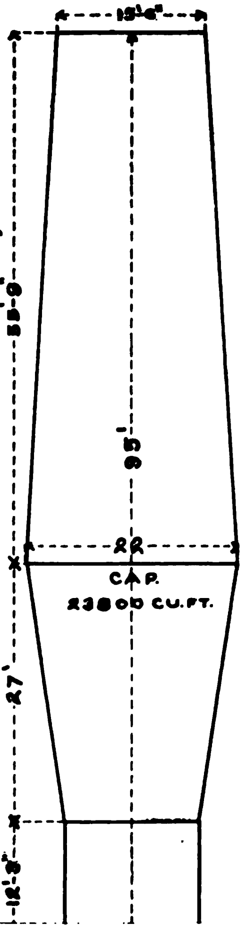

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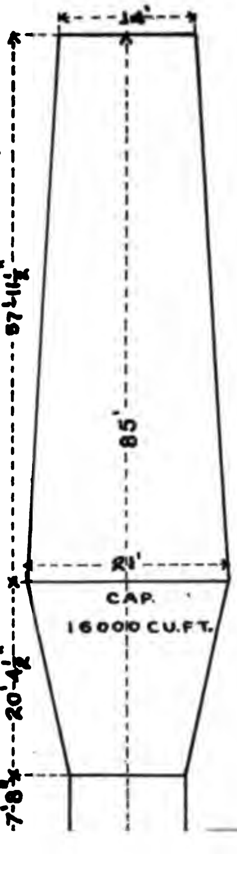

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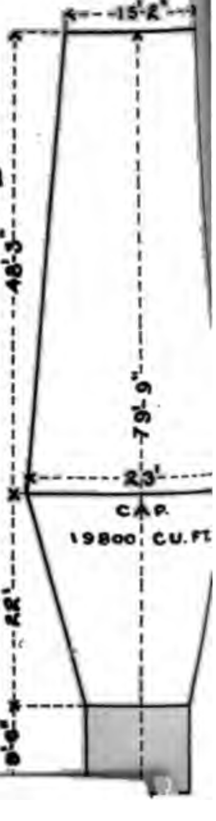

being equal, on cubic content. It does not, however, increase directly as the capacity, but in a constantly diminishing ratio to it. The consumption of fuel decreases as the capacity of the furnace increases. A similarly diminishing ratio occurs between the two, and a point is soon reached beyond which there is little or no advantage in increasing the internal space. Charcoal blast furnaces have a capacity of 4,000 to 6,000 (say 5,000) cubic feet. Coke blast furnaces are of not less than 10,500 cubic feet, and as a rule not over 21,000 cubic feet, although some modern huge furnaces show higher figures — the Duquesne (Pennsylvania) furnace, *e.g.*, 25,000 cubic feet; the Lorain furnace (Ohio), 27,420 cubic feet; the Youngstown (Ohio), 28,175. (Let 20,000 cubic feet be an average.) While at the beginning of the nineties 90 cubic feet[1] were found necessary to produce 1 ton pig iron per day, at present this has been reduced to 51.5 cubic feet,[2] or perhaps still less, by the change in the profile (or lines) of the furnace and the manner of working.[3]

(3) *Ratio of Height and Bosh Line.* High and narrow furnaces offer much resistance to the descent of the charges and to the ascent of the gases. Low and wide furnaces cause the charges to descend unevenly, as in the centre, where there is little friction as compared with that at the walls; the heavy parts of the charge will accumulate and descend in advance of the coke, which is forced against the wall. The tendency of the gases to rise near the wall is fostered; they pass through the coke, where they are not wanted, and not through the ore. The height is usually three to four times as great as the diameter of the bosh line; with the fast driving so common at present it has been made four to six times as great. Charcoal furnaces ten years ago were 48 feet high; they now reach 60 feet. Coke furnaces which six years ago were 70, 80, 85, and perhaps 90 feet high have reached 106½ feet. In view of the fact that such high furnaces have not given the saving of fuel that was expected, the tendency at present (1904) is to limit the height to 90 feet.[4] Anthracite furnaces are about 75 feet high. The distance of bosh line above the bottom of the hearth is one-third to one-half of the total height. In the older coke furnaces the bosh begins a few feet above the level of the tuyères. In modern furnaces it usually coincides with the tuyère line. Any vertical extension of the hearth further upward is liable to be very quickly eaten away.

(4) *Ratio of Stock and Bosh Line.* Extremes are from 1 : 1 with a cylindrical furnace to 1 : 2 with a much-contracted throat. The common ratio lies between 3 : 4 and 5 : 6.

(5) *The Angle of Bosh* is not under 65° from the horizontal. It lies commonly between 70° and 80°, and is usually about 75°.

(6) *Diameter and Depth of Hearth.* With charcoal furnaces the hearth is 6–8 feet in diameter; with coke and anthracite furnaces usually 8–10 feet; but this has been increased with coke furnaces to 15 feet. The depth of the hearth is usually one-tenth to one-eighth of the total height. With some modern very high furnaces it is one-thirteenth. The ratio of the diameter of the hearth to the bosh line shows a range of from $\frac{1}{1.17}$ to $\frac{1}{2.5}$.

[1] Gayley, "Transactions American Institute of Mining Engineers," XIX, p. 956.

[2] Whiting, *op. cit.*, XX, p. 287.

[3] *United States:* Walsh, "Transactions American Institute of Mining Engineers," XV, p. 421, footnote; XVII, p. 754; Gayley, XIX, p. 956; Potter, XXIII, pp. 370, 577. *Germany:* Lürmann, "Stahl und Eisen," 1896, p. 801. *England:* "Stahl und Eisen," 1897, pp. 204, 309.

[4] Reduction of height of Youngstown, Ohio, furnaces, "Iron Trade Review," April 14, 1904.

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ASTOR, LENOX AND
TILDEN FOUNDATIONS

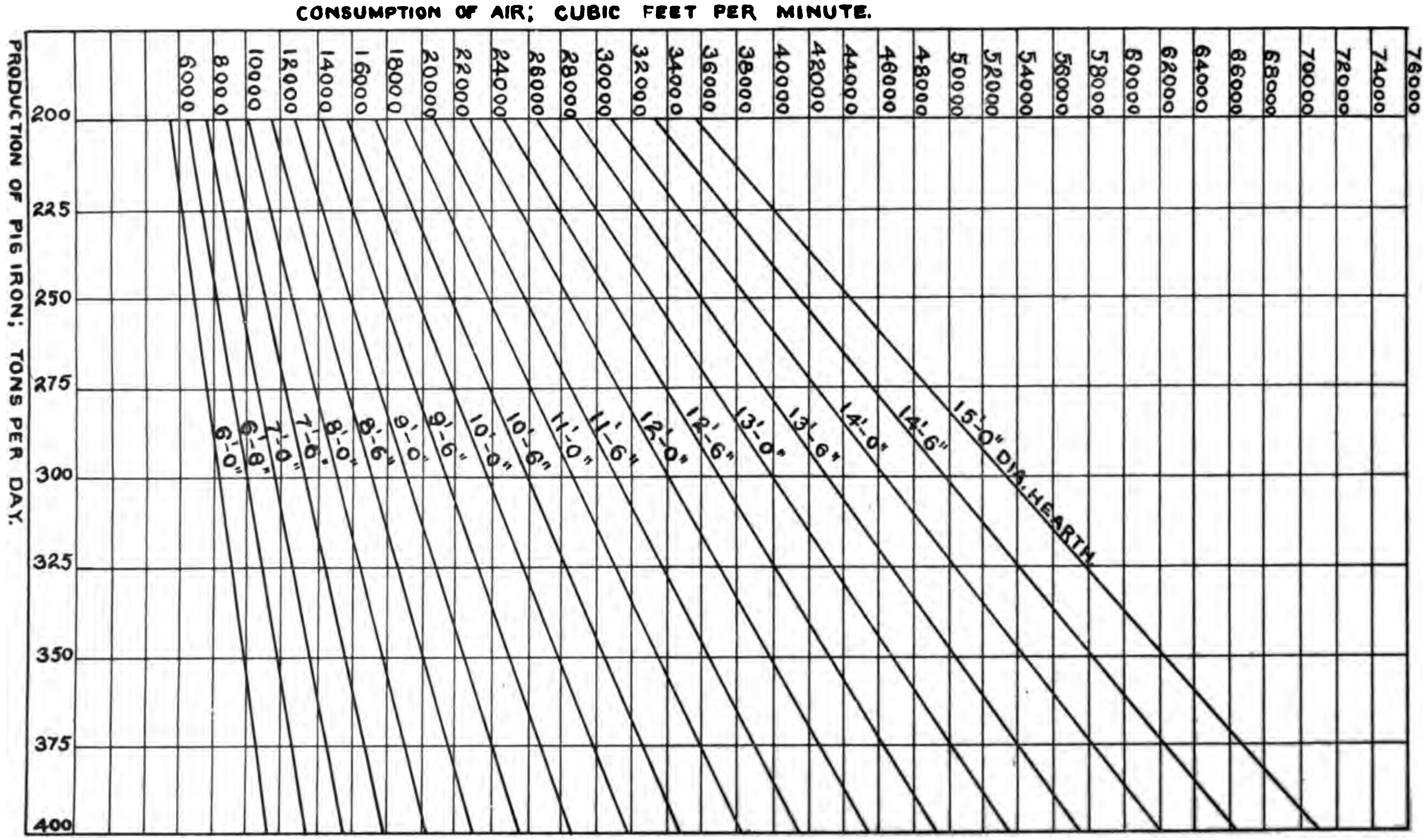

FIG. 37.

THEORETICAL DIAMETER OF HEARTH

FOR GIVEN DAILY PRODUCTION OF PIG IRON

AND CONSUMPTION CUBIC FEET OF AIR PER MINUTE.

(7) *Diameter of Hearth, Production of Pig Iron, and Volume of Air.* Assuming the ies of the furnace, the character of the ore and the coke, to be about the same, the producing wer of a blast furnace depends upon the diameter of the hearth and the volume of air blown a given time. In Figure 37 are plotted the data representing American practice with ores eraging 56.5 per cent. Fe, and with Connellsville coke.

(8) *Tuyère Line and Number of Tuyères.* The height of the tuyères above the bottom the hearth is with charcoal furnaces 4–5 feet; with coke furnaces 6–8 feet; the largest tance probably is 8 feet 2 inches.

With charcoal furnaces the number of tuyères ranges from five to eight, with anthracite and ce furnaces from eight to twelve; the diameter of tuyères is 4 inches for charcoal, 5 inches anthracite, and 6–7 inches for coke furnaces. The number of tuyères with coke furnaces been increased in some instances[1] to as many as sixteen and even twenty. The idea of a ge number of tuyères is to divide the blast more evenly, and to prevent unfused masses from lecting on the walls between the tuyères. Experience with such a large number has not, vever, been satisfactory, as it has been connected with a decrease in output,[2] and most plants re returned to their former smaller number.

(9) *Cinder Notch and Metal Tap.* The cinder notch lies 2–3½ feet below the tuyère line, metal tap 3–5 feet below cinder notch.

SECTION 10. **Construction of Blast Furnaces.**[3] (1) *Foundation.* On account of the mendous pressure exerted on a comparatively small area (for a filled furnace about 120 pounds r square foot of hearth area for each running foot of height), the foundations of a blast furnace ist be absolutely secure. They are usually of concrete, and reach well outside of the base tes of the column supporting the shaft. They vary greatly in depth with the character of e ground and the height necessary for discharging the slag from the slag runner into slag rs. Foundations, usually, are not less than 10 feet deep. The top, of concrete, is followed courses of "squares" (bricks usually 13½, 11, and 9 inches long, each 4½ x 2½, set on end) me 30 inches in thickness. One of the largest foundations erected is that of the Duquesne rks. (See Figure 32.) The ground being swampy, piles 35 feet long were driven to bed ck. They were sawed off beneath low-water line to prevent decay, and a grill work of rails is placed upon them; then followed a concrete foundation of abouf 72 feet in diameter at the ttom and 40 feet at the top, and about 45 feet high.

(2) *Hearth and Boshes.* The bottom of the hearth is commonly made of three courses wedge-shaped fire bricks (cone-bottom blocks) 18 inches long; the sides (crucible walls) are —36 inches and the boshes 24–27 inches thick, and are usually made of fire brick. The arth and boshes are usually of Benezet brick, the parts higher up of lower-grade material, g., Clarion brick. In some instances carbon brick (coke and tar burned in retorts), or carbon-ay, and coke-clay brick have been used with satisfactory results in the hearth walls.[4]

The hearth is enclosed by a cast-iron or steel hearth jacket, and water-cooled. Figure 38 ows a cast-iron crucible jacket cooled by water flowing through a wrought-iron pipe around

[1] "Transactions American Institute of Mining Engineers," XXVIII, pp. 370, 666, 858.

[2] Grammer, *op. cit.*, October, 1903.

[3] Pavloff, Atlas.

[4] "Transactions American Institute of Mining Engineers," XXI, p. 114; XVI, p. 185; Jung, "Zeitsch. d. Vereins utsch. Ingenieure," 1891, p. 10911; Hofman, "Mineral Industry," II, p. 435.

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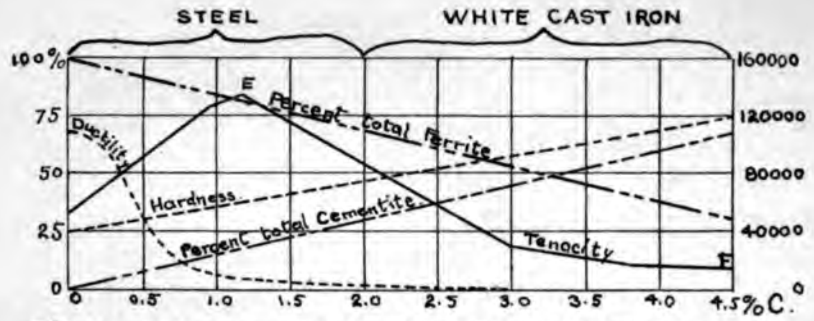


FIG. 24. EFFECT OF TOTAL FERRITE AND CEMENTITE UPON THE PHYSICAL PROPERTIES OF SLOWLY COOLED IRON-CARBON ALLOYS.

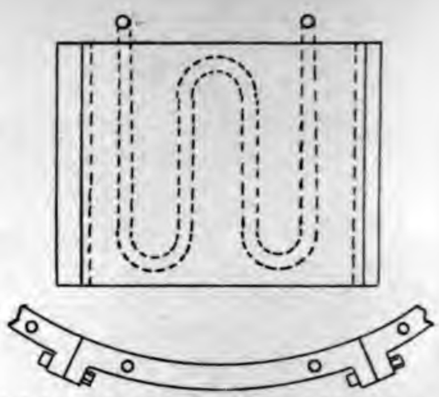


FIG. 38. CAST IRON WATER-COOLED CRUCIBLE JACKET.

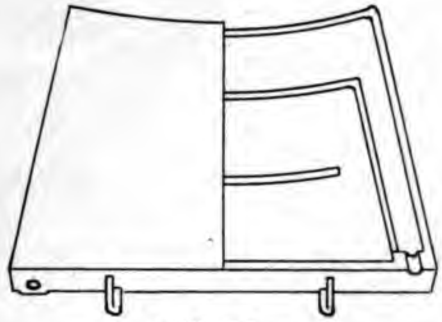


FIG. 42.

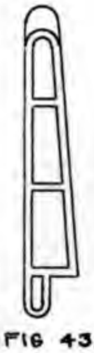


FIG 43.

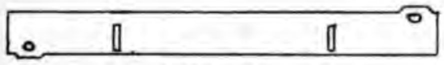
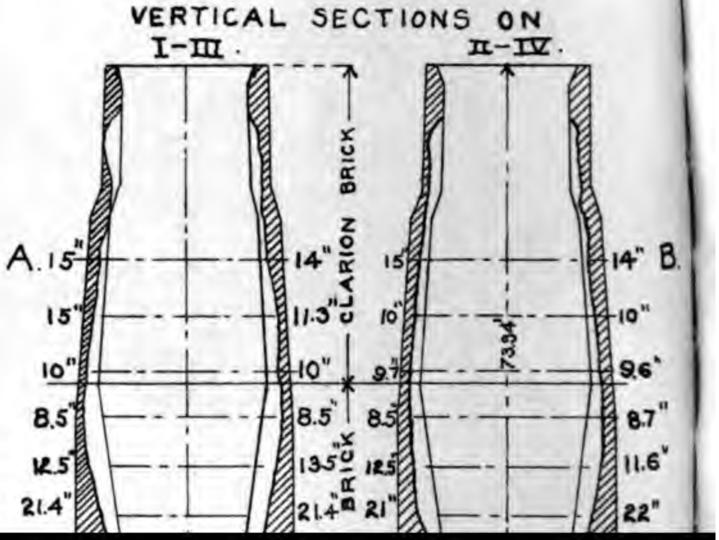
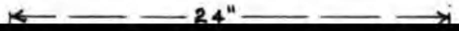


FIG. 44
SCOTT BOSH PLATES.





which the jacket was cast. The jackets encircling the crucible are bound together by strong bolts. Heavy steel plate, $\frac{1}{2}$–1 inch thick, has replaced the cast-iron jacket in most modern furnaces. Between the hearth jacket and the brickwork there used to be left an open space, filled with pebbles or broken fire brick. Cooling water was conducted through pipes to the bottom, and, overflowing at the top, was collected in a gutter surrounding the furnace, whence it ran off. Thus the lower side of the crucible, where break-throughs are liable to occur, was well protected by cooling. In recent furnaces this filling-space is omitted; the crucible wall is built against the steel plate without leaving any space for expansion. The water flows outside of the casing as usual. The cooling water is run as cold as possible.

The brickwork of the bosh walls is firmly held together by strong iron bands (see Figure 45). In order to protect it against heat and chemical action[1] it is water-cooled. At first the walls were sheathed and water sprayed against them. These spray jackets are still in use at prominent plants: Pennsylvania Steel Company (Harrisburg), Maryland Steel Company (Baltimore), Lackawanna Steel Company (Buffalo). Later 1-inch wrought-iron pipes, through which water flowed, were placed between the sheathing and the brickwork. While these arrangements, by cooling the bricks, retarded their corrosion, they did not always maintain the lines of the furnace. This was done by the use of bosh plates (water-cooled iron, copper, or bronze boxes) let into the bosh wall, which made the sheathing unnecessary.[2] The effect of the bosh plates is well shown in Figures 38–41, representing the lines of the Isabella blast furnace, Pittsburg, Pennsylvania, before blowing in, and after having been in blast for five years. The brickwork not protected by bosh plates is more or less eaten out, and even between the bosh plates it is slightly slagged away. The lining of the Edgar Thomson furnace, *H* (90 feet high, 20-foot bosh, blown in, March, 1890), lasted nine years and four months.[3]

Brickwork is worn out[4] by the attrition of the descending charge, by the chemical action of cyanogen and cyanogen salts[5] in the gases, by the action of sodium chloride in the coke, by the corrosive action of the slag, and by flaking, owing to the deposition of carbon. A blown-down furnace is usually lined with a carbon coating resembling kish. An analysis of furnace graphite gave[6] Fe, 17.3; SiO_2, 2.20; C, 70–74; while the native graphite contains Fe, 1.6–4.6; SiO_2, 2.4–8.6; C, 78–85. The older bosh plates[7] are now being replaced by the Scott removable plates (Figures 42-44).[8] The location of the plates and the manner of supplying them with water, as designed by Bauman, is shown in Figure 45.[9] The water passes from the main supply pipe, *A*, to the inclined pipes, *B*, feeding the annular supply pipes, *C*, placed in the centre of the horizontal bands, *D*, from which they are suspended by band brackets. Since

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[1] Gayley, "Transactions American Institute of Mining Engineers," XXI, p. 102.

[2] "Journal Iron and Steel Institute," 1901, I, p. 236; "Iron Age," February 14, 21, 1901.

[3] "Iron Trade Review," April 2, 1903, p. 72.

[4] Lürmann, "Stahl und Eisen," 1892, p. 122; 1898, p. 168; "Engineering and Mining Journal," June 23, 1894, p. 586; Osann, "Stahl und Eisen," 1903, pp. 336, 775, 823; Firmstone, "Transactions American Institute of Mining Engineers," October, 1903.

[5] "Berg- und Hüttenmännische Zeitung," 1904, p. 184.

[6] Cook, "School of Mines Quarterly," XIX, p. 240.

[7] "Transactions American Institute of Mining Engineers," XXI, p. 102; "Iron Age," July 2, 1891.

[8] "American Manufacturer," September 3, 1897.

[9] "Iron Age," July 22, 1897.

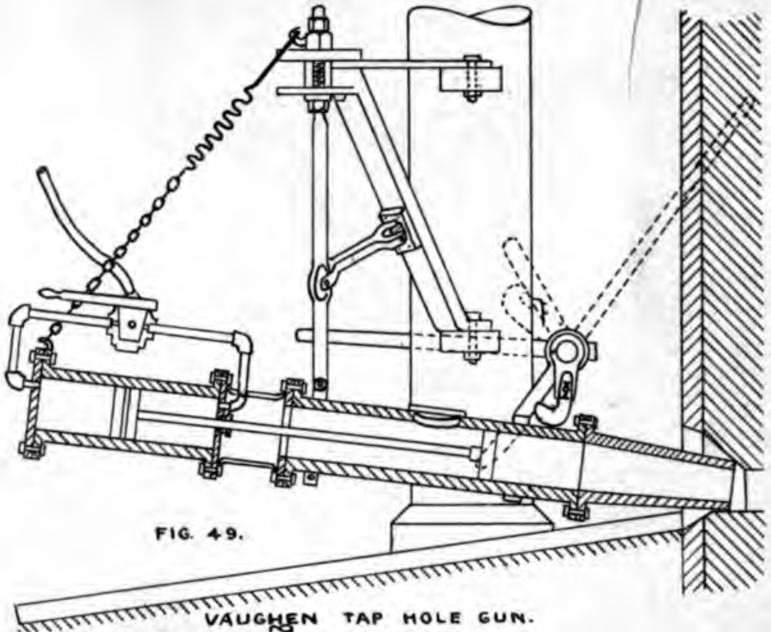
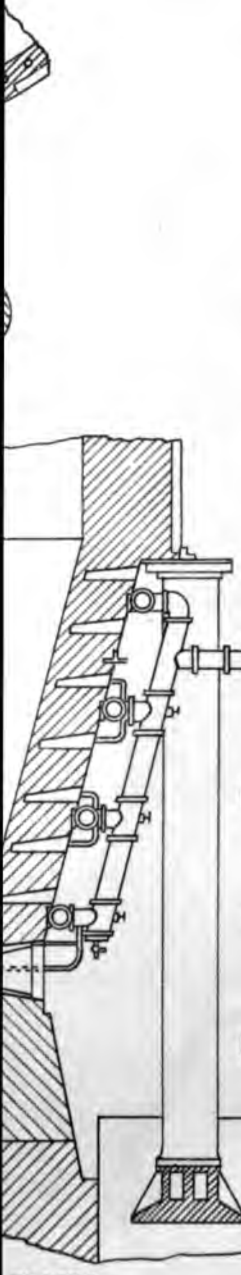


FIG. 49.

VAUGHEN TAP HOLE GUN.



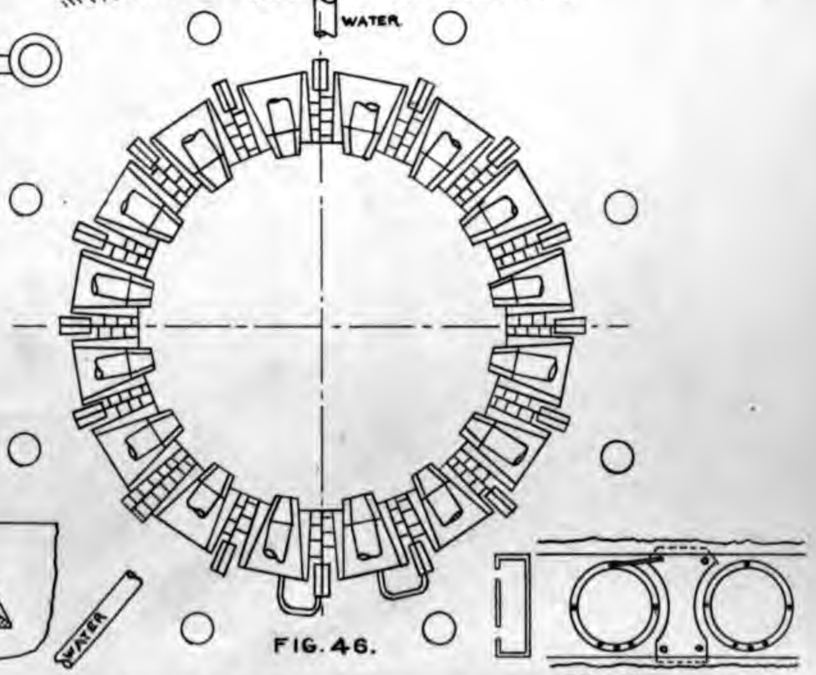


FIG. 46.

FIG. 47

PLATE 5.

SCOTT COMBINED COOLER AND BUCKSTAVE.

the number of tuyères in large furnaces has been increased from seven to as many as sixteen, it has become necessary to strengthen the brickwork between the tuyères if break-outs are to be avoided. The Scott combined cooler and buckstave (Figures 46–47)[1] does this satisfactorily. It consists of a bronze casting held in position by the band above the tuyères and by the hearth plate. It carries the weight of the brickwork from the hearth plate to the mantle plate, and thus takes away the pressure from the tuyère holders and makes them easily removable.

The bosh wall is seen (Figure 36) to merge into the fire brick in-wall (69) of the trunk. This is followed by the in-wall backing (70) of second-class fire brick, the expansion space (75) filled with sand or crushed stone, the in-walls filling (71) of red brick, and the iron sheathing (46); the greatest thickness of the shaft wall reaching 40 inches. The trunk rests on the mantle beams (5), which are bent to the circle of the furnace and are supported by eight to ten hollow cast-iron columns 20–28 feet high. Their great height makes the boshes readily accessible, easy to cool and to watch.

(3) *Metal Tap.* All modern furnaces have a closed front, or internal crucible, which means that metal and slag collect in the crucible, forming the continuation of the bosh, and are tapped at intervals. The tapping-hole channel has a slight rise from the bottom of the crucible outward in order that there may always remain a bath of metal in the furnace, which will, if not prevent, at least somewhat hinder particles of charge from adhering to the bottom. The opening of the tap hole is often attended with difficulty when the tap has become hard. Baker has introduced a modified steam percussion drill (Figure 48)[2] to open the tapping hole. It is mounted on a frame that can be swung into place in front of the tapping hole, and will open it in from three to five minutes, which by hand drilling sometimes takes a crew of eight to ten men ten to thirty minutes and longer. Grammer[3] constructed an electric burner to open hard taps. Several devices have been suggested for a continuous flow of iron and of slag from the furnace,[4] but have not yet come into use. The usual way of plugging the tapping hole after the metal has flowed out is to shut off the blast, and then ram in balls of fire clay mixed with crushed brick, graphite, old plumbago crucibles, etc. The stopping of the blast causes delay and cooling of the furnace, and the closing of the iron notch by hand is slow, hard work, and liable to be accompanied by accidents. Most large furnaces use the Vaughan tap-hole-closing machine (tap-hole gun) (Figure 49).[5] It consists of a steam, or, better, air cylinder *B* (2 feet $11\frac{3}{4}$ inches long and 11-inch bore, with steam-inlet pipes, *H* and *H'*, and four-way cock *F*), joined by the yoke *D* ($11\frac{7}{8}$ inches long), to the main cylinder *A* (2 feet $7\frac{7}{8}$ inches long and $7\frac{7}{8}$-inch bore), ending in a nozzle, *C* (18 inches long, contracted to 6-inch bore). The gun is supported by the iron rod *M* and the crane *W*, *X*, *Y* (the lugs of which are fixed to one of the blast-furnace columns), and swung into place by the lever *W'*, having its fulcrum on the shaft *O*. Clay piston *A'* and steam piston *B'* are connected by the rod *r*. The working cylinder *A* has on the upper side a slot, *S* ($4\frac{1}{2}$ inches wide x 8 inches long), to

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[1] "Iron Age," September 2, 1897, p. 12.

[2] "Transactions American Institute of Mining Engineers," XXI, p. 590.

[3] *Op. cit.*, 1901, XXXI, p. 626.

[4] Stapf, "Iron Age," November 12, December 10, 1903; "Stahl und Eisen," 1903, p. 1344; Lattmann, "Stahl und Eisen," 1903, p. 1224; Bratke, *op. cit.*, 1903, pp. 1033, 1083, 1345.

[5] "Iron Age," November 21, 1895; "Transactions American Institute of Mining Engineers," XXVII, p. 40; "Stahl und Eisen," 1901, p. 1098.

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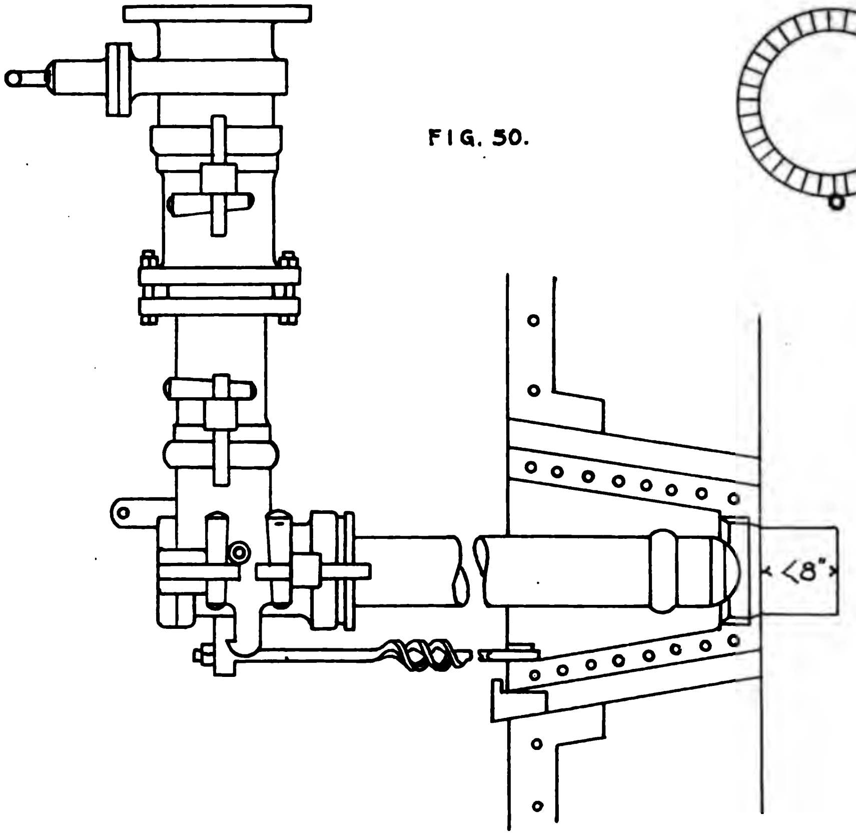
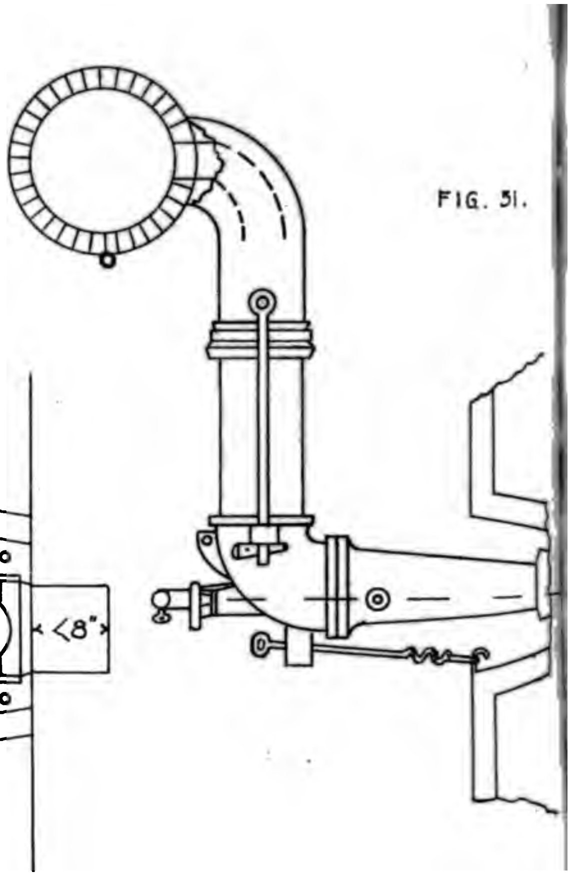
FIG. 50.



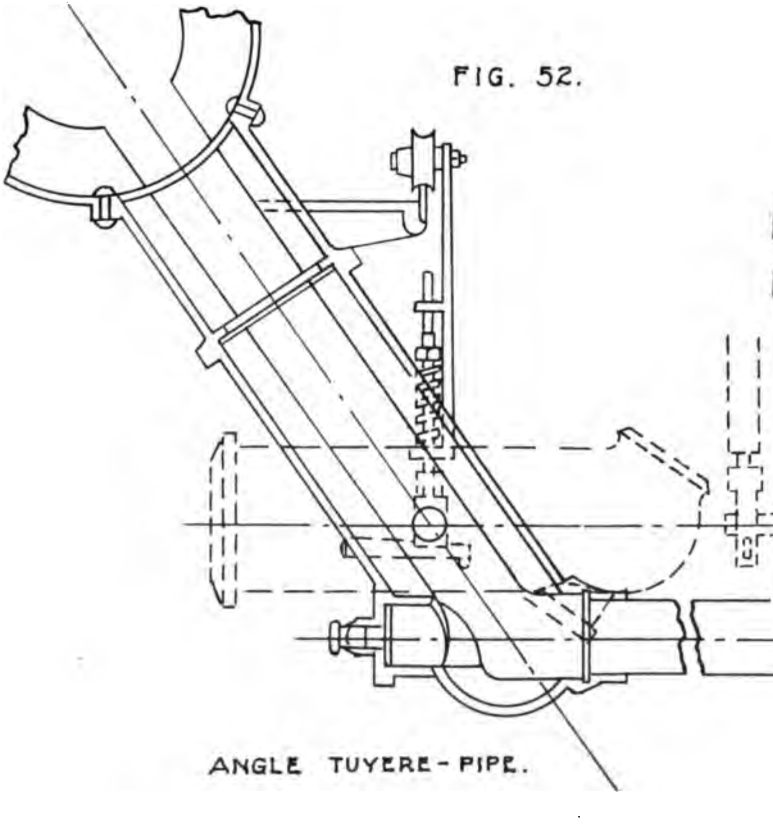
PLAIN ADJUSTABLE TUYERE-PIPE.

FIG. 51.



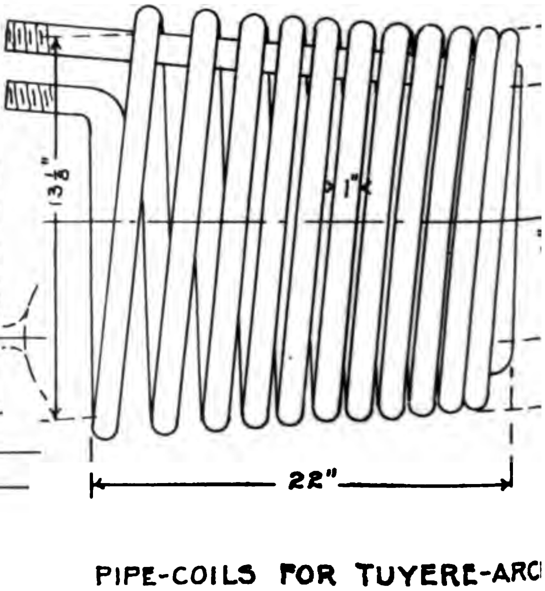
TUYERE-STOCK OF THE
MARYLAND STEEL CO.

FIG. 52.



ANGLE TUYERE-PIPE.

FIG. 53.



PIPE-COILS FOR TUYERE-ARC

receive the clay balls. In operating, the gun, filled with clay balls, is swung into working position when the metal has begun to flow, and is held there by the lever *N*, ready to push the clay balls into the tapping hole just before the metal ceases to flow. With a new furnace a single charge of the gun is sufficient; as a rule the working cylinder has to be filled two to three times. The advantages of the device are that no time is lost, the blast need not be stopped, and there is next to no danger. Hartmann[1] proposes to use a graphite-clay cylinder in the iron notch, to ram into it a graphite-clay mixture, and to force into this an iron bar after the notch has been closed. The bar is to be withdrawn later on, and the tap hole kept cool, by blast if necessary. In tapping, only a thin shell will have to be pierced, and the tap hole will keep its size, permitting the iron to flow in a thin stream and preventing it from coming out with a rush.

(4) *Cinder Notch.* All blast furnaces at present use some form of water-cooled casting with hole for tapping the slag. Lürmann, in 1867,[2] invented the first tapping jacket or cinder notch. A great variety of forms are in use. The most common one in this country is the one shown in Figure 50. It had its origin at Durham furnace[3] in 1876, and is made in three parts in order to facilitate its removal. These are the iron cinder breast *A*, having a height equal to the thickness of the crucible wall; the cinder block *B*, made of brass, reaching in the crucible; and the cinder notch *C*, or monkey, made of silicon copper, reaching in still further. *E*, *D*, and *F* are the water feed pipes. The outlet pipes are not shown. The trough, lined with raw and burned clay, reaches into the cinder breast and protects it from the slag. The monkey is protected by the slag that chills around it.

(5) *Tuyère Stock.* Tuyère nozzles are usually 6 inches in diameter with coke furnaces, 5 inches with anthracite, 4–5 inches with charcoal furnaces. Figures 50, 51, and 52 represent three forms of tuyère pipes. Figure 50 is a plain, adjustable tuyère pipe, made by Taws and Hartman, Philadelphia. The tuyère is made of bronze, with water circulating in the annular space. It reaches several inches into the furnace, thus allowing the furnace to have a larger diameter of hearth without any increase of pressure of blast, and at the same time protecting the walls. The distance that the tuyères reach into the furnace[4] depends on the pressure of blast and diameter of hearth. A common figure is 5 inches. The tuyère arch or tuyère breast is a pipe coil, say 1 inch in diameter (Figure 53), around which is cast cast iron. The cooling-water ought to be run as cold as possible. The different parts of the tuyère are usually of cast iron. It is essential that they shall permit of being easily taken apart, hence they are held together by key bolts. Figure 52[5] represents the tuyère stock that is used at the furnaces of the Maryland Steel Company (1894); it is one that is very common where a vertical tuyère pipe is used.

Figure 53[6] shows a construction by Philadelphia Engineering Works of a tuyère pipe that is inclined, which facilitates its being turned down and out of the way; the bearings of the

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[1] "Iron Trade Review," November 21, 1901.

[2] "Stahl und Eisen," 1887, p. 789; 1891, p. 553.

[3] "Iron Trade Review," November 21, 1901, p. 341.

[4] Cochrane, "Stahl und Eisen," 1882, pp. 434, 555.

[5] Richards, "Notes on Iron."

[6] "Iron Age," June 8, 1893.

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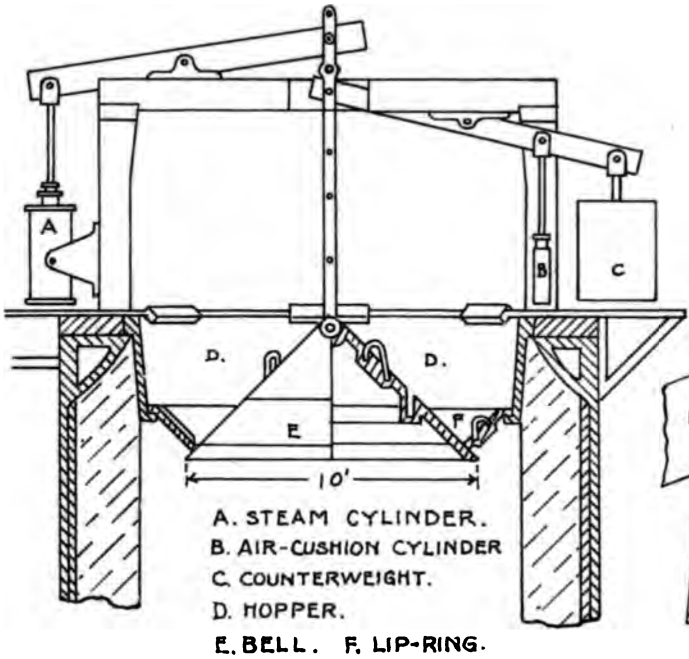
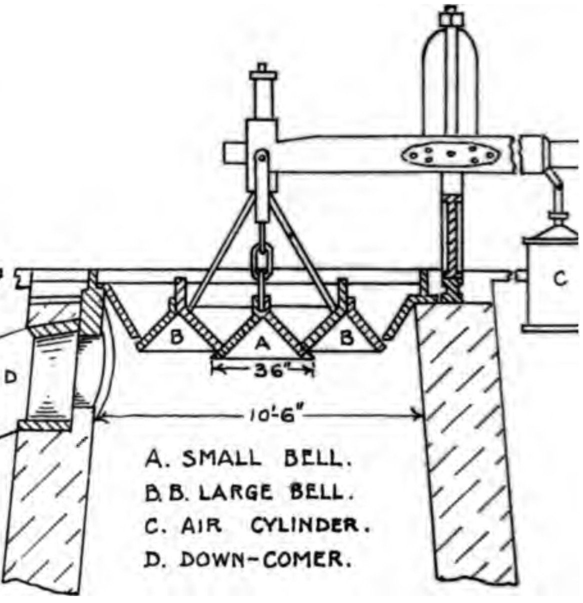
FIG. 55.



SINGLE BELL AND HOPPER.

FIG. 56.



DOUBLE BELL AND HOPPER.

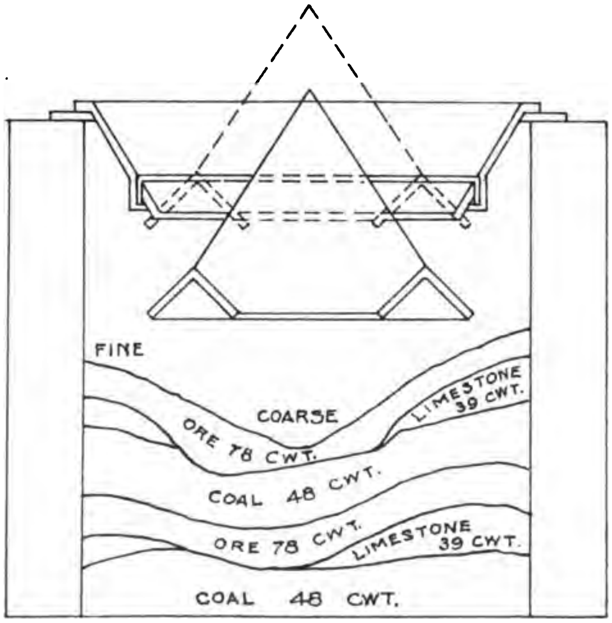


FIG. 57.

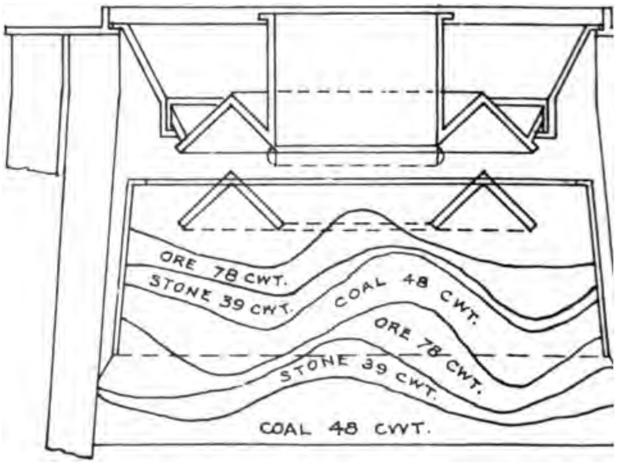


FIG. 58.

DISTRIBUTION OF CHARGE.

trunnions on the sides rest in bearings swung from two spring balls suspended from a carriage playing on a horizontal bar.

(6) *Feeding of Blast Furnace; Withdrawal, Purification, and Burning of Gases.*[1] Blast furnaces formerly had an open top; the gases (60N, 24CO, $12CO_2$, H 2, CH_4) were burned there, and their fuel value mostly lost. At present all furnaces, in this country at least, have a closed top. The gases, issuing with a pressure of $\frac{3}{8}$–$\frac{3}{4}$ inch H_2O, are drawn off at the top of the charge and conducted to stoves, boilers, or other places where they are to be burned. In some European works the top of the furnace remains open while the gases are drawn off some distance below the top of the charge.

Single Bell and Hopper. The first bell and hopper, invented by Parry, in England, in 1850, has been considerably modified. In the illustration (Figure 55) the cast-iron bell with the lip ring is suspended by a bell rod and a double-lever system. The bell is cast in one or two pieces, according to the size, and the lip ring always in segments. One of the levers carries at the free end the counterweight of the bell, and to that of the other is attached the steam or air cylinder for lowering and raising the bell. The two are connected by a link to the centre of which is pivoted the bell rod. The bell is lowered concentrically, thus insuring a regular and even distribution of the stock (ore, flux, and fuel). With the hand-barrow system the stock is brought in cars to the mouth of the hopper and dumped, and then the bell is lowered. As a rule, the fuel is dumped and lowered by itself, to be followed by ore and flux together. There is usually an iron basket with burning coal near the hopper to ignite the gas issuing forth when the bell is lowered.

Double Bell and Hopper (Figure 56).[2] The use of the two bells, *A* and *B*, makes it possible to feed part of the charge more toward the centre by lowering first bell, *A*, and the rest toward the periphery by lowering *A* and *B* together.

From the investigations of Firmstone,[3] the Figures 57–58 are selected from a large number to show the effects of feeding with a single and double bell. For other feeding devices see Steger, "Zeitschrift für Berg-, Hütten-, und Salinen-Wesen," 1902, L, page 97. Feeding with the usual bell and hopper presupposes the hand-barrow system; the charge is brought to the feed floor in buggies, which are pushed by the top-fillers to the hopper and then dumped. A furnace putting through in twenty-four hours say seventy charges weighing 3,000 pounds (1,500 ore, 600 flux, 900 coke) can be managed by two top-fillers. This is not possible with a furnace treating seventy charges that weigh 53,000 pounds (33,000 ore, 6,000 flux, 14,000 coke), as is, *e.g.*, the case with the Duquesne, Eliza, and other furnaces; mechanical feeding, therefore, becomes necessary. In the latest arrangements for mechanical feeding, the amount of gas lost in lowering the bell (a very important item[4]) has been greatly reduced by having a double set of bells and hoppers, one above the other.

The Neeland Charging Apparatus of the Duquesne Furnace Plant is shown in Figures 59–60.[5] It consists essentially of three parts: (1) the main bell, *p* (with its hopper), suspended

[1] Lürmann, "Stahl und Eisen," 1899, p. 771; footnote, list of references.

[2] "Transactions American Institute of Mining Engineers," XIV, p. 130; XVIII, p. 390, Plate II, Figure 8.

[3] *Op. cit.*, XXVIII, p. 370.

[4] Simmersbach, "Stahl und Eisen," 1898, p. 890.

[5] "Iron Age," March 25, 1897, p. 8.

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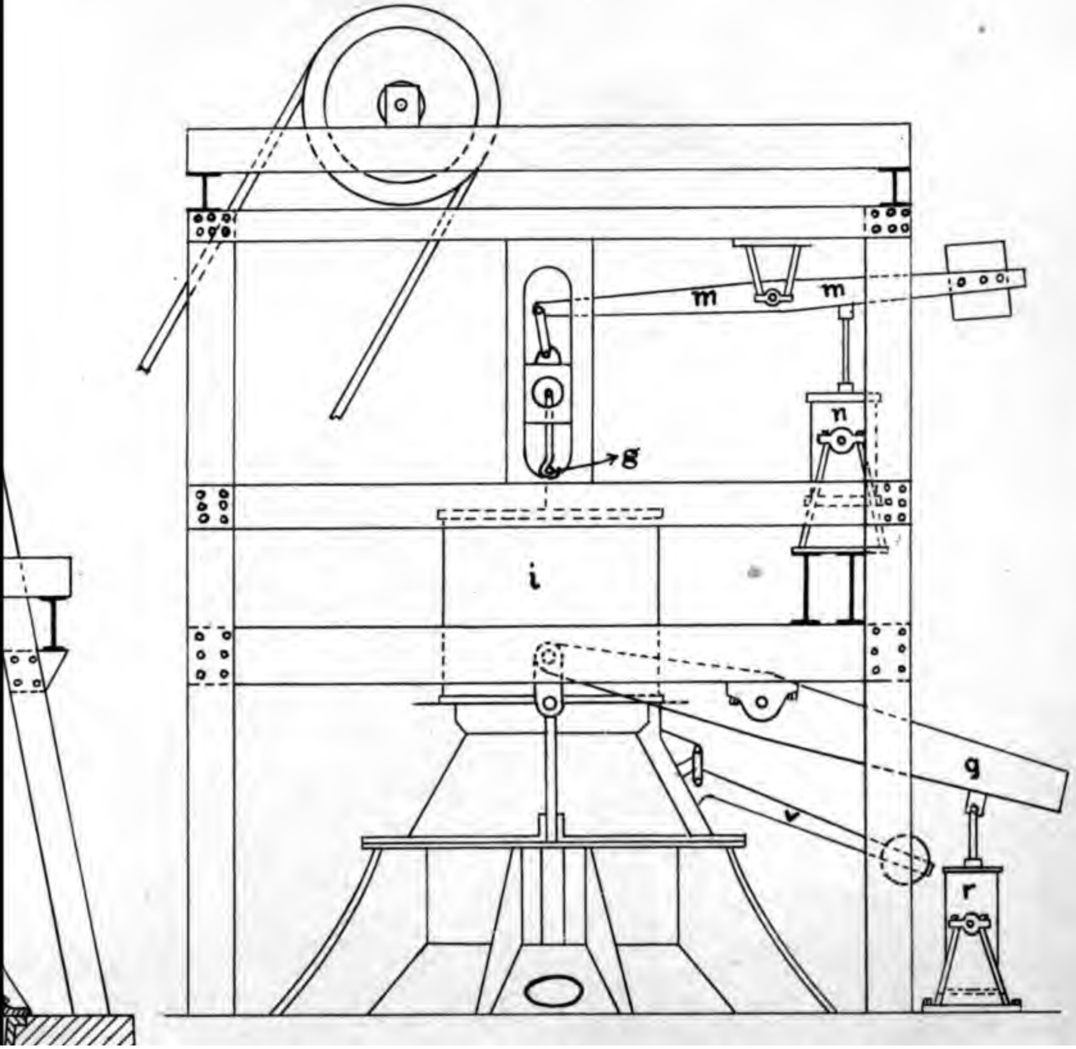
m
m
n
g
i
g
v
r

by two bell rods from the cross head, and this by two rods from a forked lever operated by a steam cylinder, r; (2) the gas-sealing bell, u, balanced by rods and levers with counterweight; (3) the charging bucket ($\frac{3}{8}$-inch steel, 5 feet 7 inches in diameter, holding 10,000 pounds ore, 4,000 pounds coke or limestone), consisting of the iron casing, i, resting on the bell-shaped bottom, which is secured by the rod h and a link to the lever m, with counterweight, and actuated by the steam cylinder n.

The bucket, having been hoisted on the incline from the ore floor (see Figure 32) to the top of the furnace, is lowered. The lower flange of the casing or shell, i, strikes the top of the hopper, t, and remains there when the bottom moves away from the casing and pushes down the gas sealing bell, when the contents of the bucket run out and are evenly distributed over the main bell, p. On its return the bottom picks up the shell, and allowing the gas sealing bell to return to its place, descends to the furnace floor to be refilled. The main bell, p, is now lowered, and the charge dropped as usual, with the exception that only that amount of gas is lost which will fill the space between the two bells. There is nobody at the top of the furnace; all the valves, etc., are controlled by the hoisting engineer below.

In order to protect the sides of the furnace from mechanical wear at the stock line they are sometimes lined with cast-iron plates.

Baker[1] criticises mechanical top-charging in regard to the distribution of the furnace and the large fuel consumption they cause. Regular Lake Superior ore, as used in most large American smelting plants, contains about 10 per cent. lump ore, and Connellsville coke is very strong. With these there is little danger of the charge becoming so unmixed as to leave on one side of the furnace solid coke, flux, and lump ore, and on the other coke dirt and fine parts of the charge. At the works of the Dominion Iron and Steel Company of Sydney, Cape Breton, using Belle Island, Newfoundland, ore, which is all lump similar to red brick, and friable coke, this happened, with the result that the furnace was burned out on one side after a few weeks' run. Special devices had to be used to insure a uniform distribution and with it a satisfactory campaign. As to the fuel consumption, mechanical top-charging necessitates at present dropping the coke six to seven times as against three to four times with the hand-barrow system. The result is breakage of coke, formation of harmful coke dirt, and consequent increase of fuel consumption.

Several mechanical top-charging devices are given on page 39. Others are those of Sahlin,[2] Brown,[3] Newman,[4] Roberts,[5] Pneumatic Lowering of Bell.[6] The stock indicator of Baker[7] and the automatic charging recorder of Neumark[8] may be mentioned in this connection.

The diameter of the bell is of importance in regard to the distribution of coarse and fine particles of the charge. It is usually made a little smaller than one-half of the diameter of the

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[1] "Transactions American Institute of Mining Engineers," February, 1904.

[2] "Journal Iron and Steel Institute," 1903, I, p. 231.

[3] "Iron Age," April 12, 1901.

[4] "Iron Trade Review," April 12, 1902.

[5] United States Patent, December 27, 1898, No. 616,494.

[6] "Compressed Air," 1903, VIII, p. 2555; "Stahl und Eisen," 1901, p. 734.

[7] "Iron Age," July 18, 1901.

[8] "Stahl und Eisen," 1902, p. 816.

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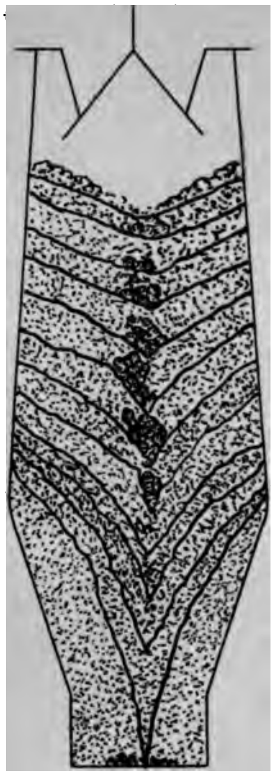

FIG. 61.

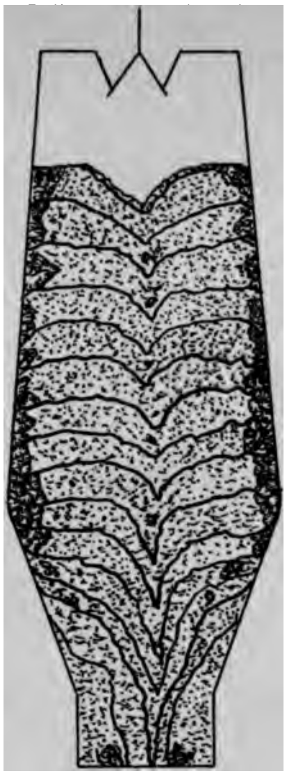

FIG. 62.

SPEARMAN KENNEDY GAS BURNER.
FOR STOVES.

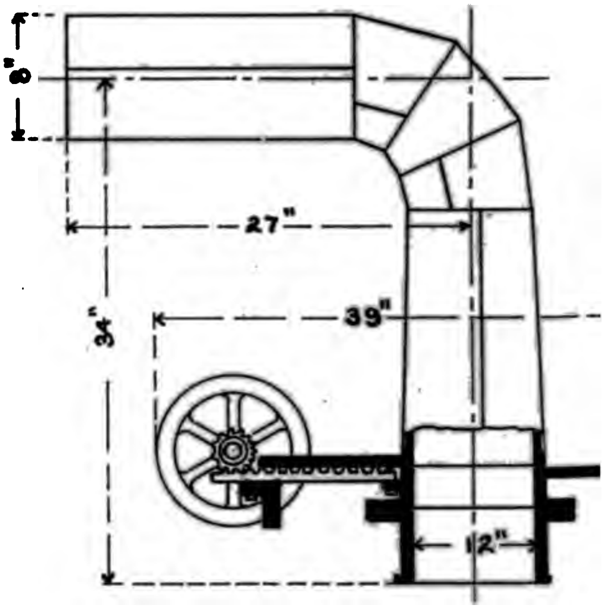

FIG. 63.

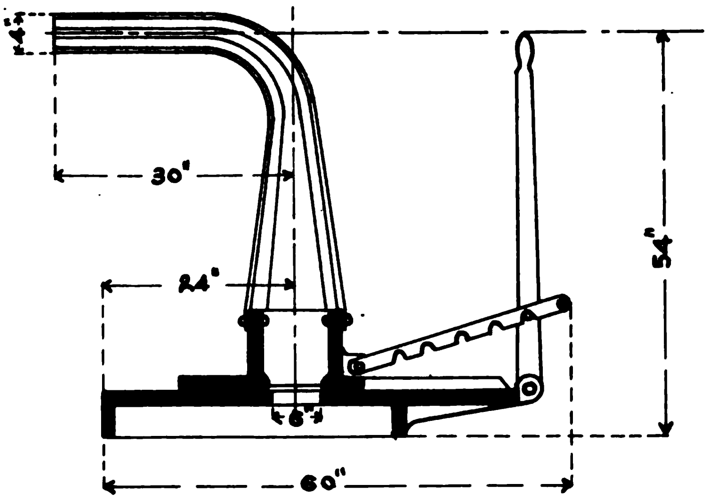

FIG. 64.

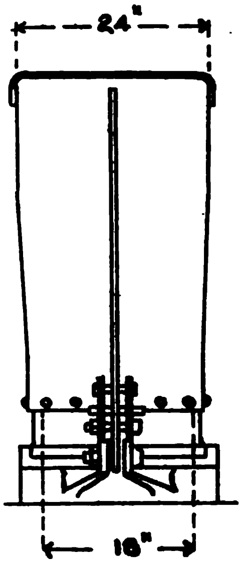

FIG. 65.

SPEARMAN KENNEDY GAS BURNER FOR BOILERS.

bosh (see blast-furnace table, facing page 45). Figures 61 and 62,[1] selected from sixteen by Richards and Lodge, who obtained them by charging gravel and sand into a model of a furnace 40 inches high (the charges being separated from each other by a layer of fine coal), withdrawing the contents at the bottom, and noting the changes in the descent. Figure 61 shows that with a large bell the lumps roll to the centre; Figure 62 that with a small one they go mainly to the circumference, although some gather near the centre. The aim in charging is to have coarse ore more toward the centre than the circumference, so as to make the gases ascend evenly through the charge, and not to allow them to rush up at the circumference, for which they have a tendency.

The downcomer (39, Figure 36) is of sheet iron. It has to be large enough to allow 72.25 cubic feet of gas at 0° C. (or $Vt = Vo(1 + 0.00366\, t)$) to pass through for every pound of fuel burned in the furnace, the usual velocity being 10 feet per second. On the top of the downcomer is the bleeder (37, Figure 36), a vertical pipe with safety valve (38), through which some gases will always pass off and burn, provided the pressure is sufficient. It thus forms an indicator of the pressure. The downcomer is always at a steep angle to prevent dust from settling out in it. It usually has explosion doors (76) and ends in a dust catcher (42) with discharge bell. The gases from the dust catcher pass off through elevated sheet-iron flues in which often provision is made for the removal of dust[2] by forming the lower side into a series of hoppers. The downcomer (39) reaches down into the dust catcher (40) so as to force the gas current to change its direction and thus drop more dust.

The blast-furnace gas is burned in stoves, under boilers, and lately it has been used directly in gas engines. When the gases are to be burned in boilers and stoves, wet condensation is not much used. Although it readily removes the water vapor of the furnace gases and to some extent also the dust, it has not found much favor, as it cools down the temperature of the gases to such an extent that the loss of the sensible heat is not made up by the gain in removing the water vapor. Water is sometimes used as a seal in the dust catcher (40, Figure 36), the lower end of which, instead of being closed by a bell (42), ends in a pipe reaching into a basin of water from which the dust is shoveled out as a fine mud. The Brier Hill gas washer of Steel and Ford,[3] however, is in operation at Youngstown, Ohio, blast furnaces, smelting charges with 90 per cent. Mesabi ores. This washer removes the coarse particles only. It has reduced the cleaning of boilers and of stoves to one-fourteenth of what was formerly necessary, while the temperature of the gases is diminished only 15° C. When gases are to be burned in gas engines wet condensation becomes a necessity. According to Grüber,[4] blast-furnace gas contains per cubic metre 12–45 grams of floating dust, and at a temperature of 150° C., about 160 grams water. The dust is now reduced to 0.01 and even 0.005 gram, and the water to 2 grams.[5] The centrifugal washers mostly used in Germany are those at Differdingen[6] and of Theisen;[7] they do good work cheaply. Scrubbers and filtering devices have proved to be too expensive.

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[1] "Transactions American Institute of Mining Engineers," XVI, p. 154.

[2] Lürmann, "Stahl und Eisen," 1896, p. 958; 1897, p. 55; 1901, pp. 433, 489.

[3] "Iron Trade Review," January 22, 1903.

[4] "Stahl und Eisen," 1904, pp. 9, 89; "Iron Age," March 17, 1904; "Iron and Steel Metallurgist," 1904, I, p. 299.

[5] See also Greiner, "Journal Iron and Steel Institute," 1901, LIX, p. 56; Osann, "Stahl und Eisen," 1902, p. 153.

[6] "Stahl und Eisen," 1901, pp. 447, 619.

[7] *Op. cit.*, 1884, p. 35; 1899, p. 57; 1900, p. 1037; 1901, p. 437; 1902, p. 371; 1904, p. 285.

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FIG. 66.

BLAST AND FUEL TABLE.

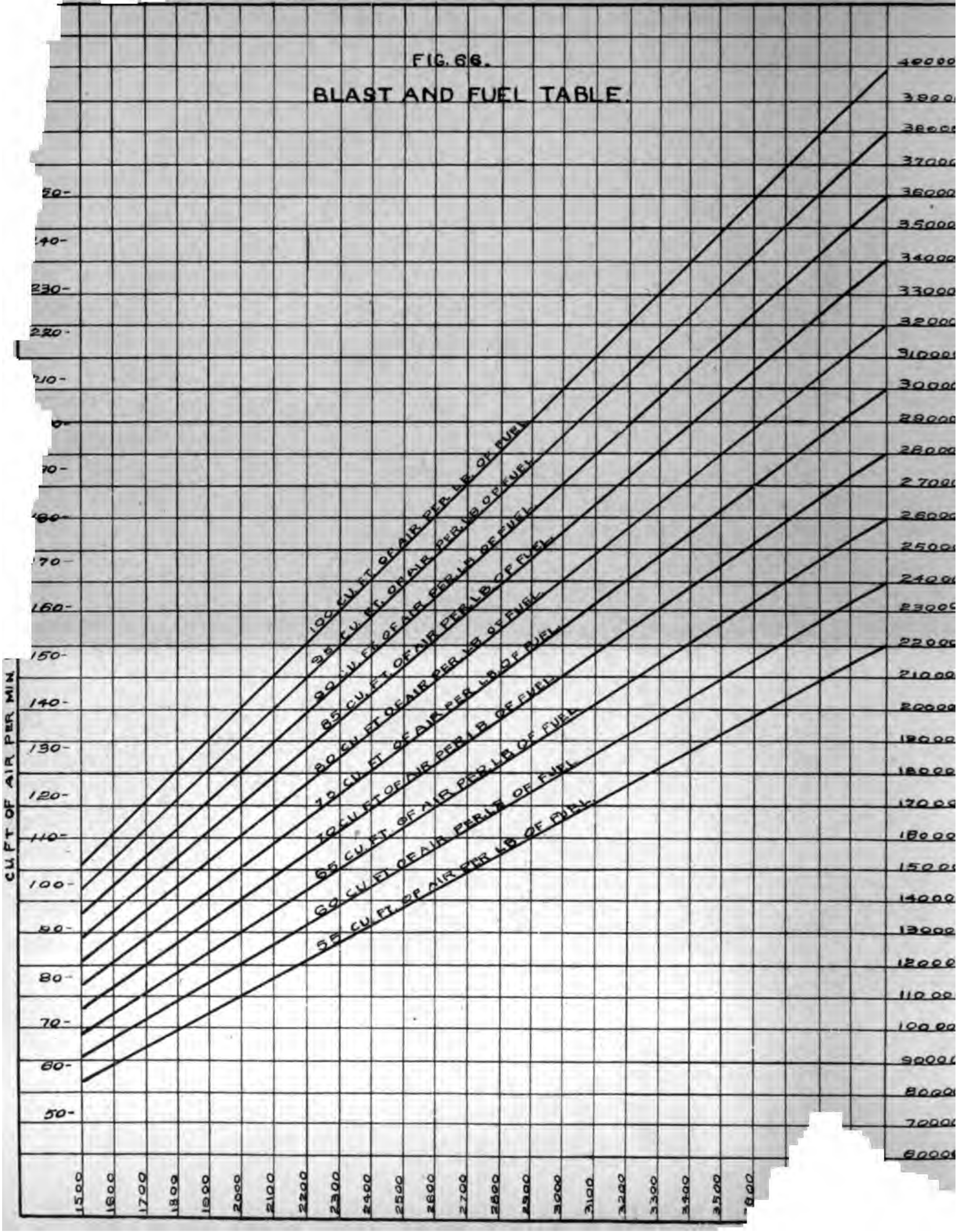

Figure 63[1] is a Spearman-Kennedy burner, with valve for a fire-brick hot-blast stove. The heavy lines represent castings machined on the top surfaces. The valve is a sliding cast-iron plate machined on the lowered surface. It is actuated by rack and pinion. Upon one end of this plate is cast a port-opening to which is attached the wrought-iron elbow. In the figure the port is open; when closed the solid part of the valve closes the port. Figures 64–65[2] represent a Spearman-Kennedy gas burner for boilers. It is built on the same general principle as the burner for hot-blast stoves, only it is wider and is moved by means of a lever and link. There are usually two burners to a boiler. The gases are burned in a large brick chamber and only the products of combustion are conducted into the boiler flues. The reason for this is the difficulty with which blast-furnace gases are ignited. They require a superheated space to insure quick ignition. A small fire is kept going in this space to reignite the gas should it for some reason or other give out for a moment. This makes explosions impossible.

SECTION 11. **The Blast.** A. *The Blowing Engine.*[3] In modern blast-furnace practice the blowing engine for a single furnace is called upon to deliver from 5,000 to 35,000 cubic feet of air per minute, at a pressure of from 8 to 25, average for coke furnaces perhaps 15, pounds per square inch. As it has to work constantly under irregular strains, with little time for examination and repair, it must be strongly built. The quantity of air required for a blast furnace can be calculated[4] from the amount of coke (carbon) that is to be burned in a given time. The amount of air delivered to a furnace can be calculated (1) from the efficiency of the blowing engine; (2) from the area of the tuyères, the temperature and pressure of the blast; and (3) from the analysis of the gases, making either the ratio $CO_2 : CO$ or the percentage of N the basis. Roughly it is estimated[5] that 5–6 tons air will yield 1 ton of pig iron; that 60–65 cubic feet air will burn 1 pound of coke, and that 220 cubic feet air will have to be delivered every minute per square foot of hearth area.[6] Figure 66 of F. C. Roberts shows the amounts of air and coke required to produce 1 ton of pig iron. At 0° C. and 29 inches barometric pressure, 1 cubic foot air $= 0.080728$ pounds, and 1 pound $= 2.387$ cubic feet.

Blowing engines have until recently been almost universally driven by steam engines, water power being used only occasionally. At present gas engines driven by the furnace gases have replaced many steam engines in Europe, instead of the gases being burned under boilers to furnish steam. In this country the Lackawanna Steel Company of Buffalo[7] has been the pioneer. Some of the leading articles on the subject are: Campbell, "Structural Steel," 1903, p. 111. Greiner, "Journal Iron and Steel Institute," 1898, I, p. 200;[8] 1900, I, p. 109; 1901, I, p. 56. Richards, "Journal Franklin Institute," 1900, CL, p. 415; "Iron Age," December 20, 1900; "Iron Trade Review," December 13, 20, 1900. Oechelhaeuser and Junkers, "Iron Age," December 6, 1900. Hubert, "Revue Universelle," 1900, L, p. 156; "Annales

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[1] "Philadelphia Engineering Works Catalogue," p. 99.

[2] *Op. cit.*, p. 112.

[3] Hauer, "Hüttenwesenmaschinen," Leipsic, 1876, 1887; Ihreng, "Die Gebläse," Berlin, 1903; J. Kennedy, "Transactions American Institute of Mining Engineers," XXII, p. 537.

[4] Ledebur-Nitze, "Engineering and Mining Journal," February 21, 1891.

[5] Cook, "School of Mines Quarterly," XIX, p. 235.

[6] F. W. Gordon, private communication, 1904.

[7] "Iron Age," January 7, 1904.

[8] Toldt, "Stahl und Eisen," 1898, p. 1141.

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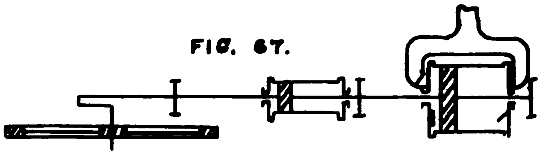
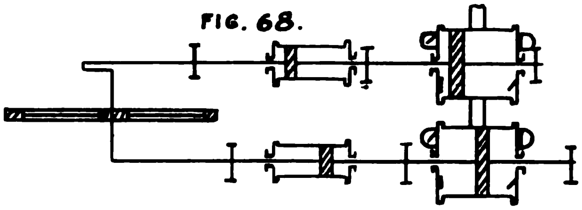
FIG. 67.



SINGLE, DIRECT, WITH FLY-WHEEL.

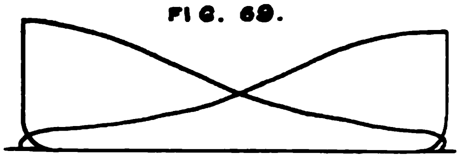
FIG. 68.



DUPLEX, DIRECT, WITH FLY-WHEEL

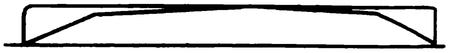
HORIZONTAL BLOWING ENGINES.

FIG. 69.



STEAM CYLINDER.

FIG. 70



AIR CYLINDER.

SINGLE ENGINE INDICATOR CARDS.

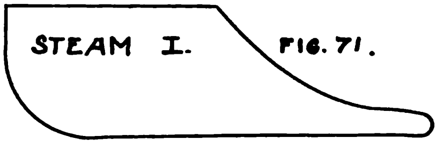
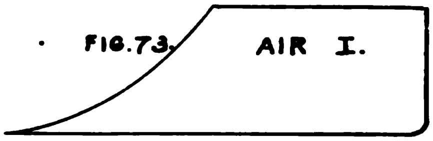


FIG. 71.

FIG. 73.



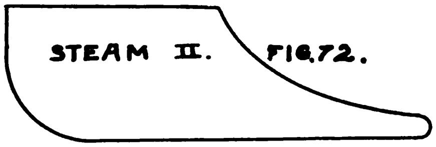
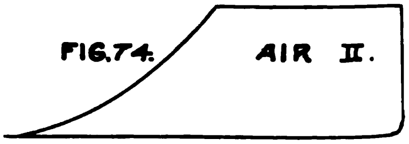


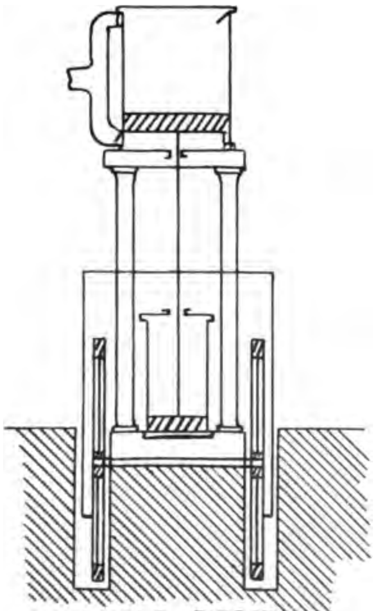
FIG. 72.

FIG. 74.



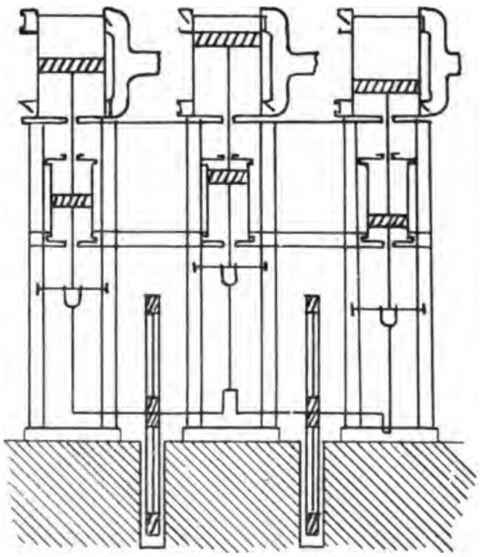
DUPLEX ENGINE INDICATOR CARDS.

FIG. 75.



SINGLE, DIRECT.

FIG. 76.



TRIPLE, DIRECT.

VERTICAL BLOWING ENGINES WITH FLY-WHEELS.

FIG. 77.

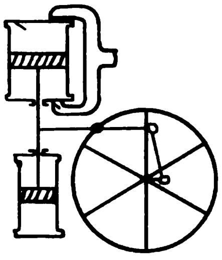
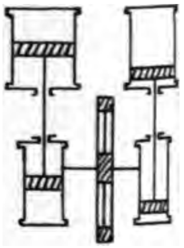
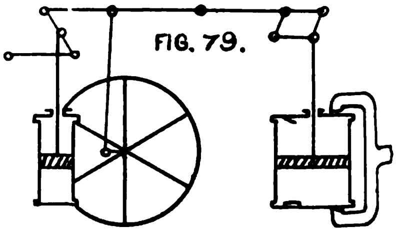


FIG. 78.



VERTICAL DUPLEX BLOWING ENGINE WITH BEAM.

FIG. 79.



VERTICAL SINGLE BLOWING ENGINE WITH BEAM.

des Mines de Belgique," 1897, II, p. 233; "Bulletin Industrie Minérale," 1901, XIV, p. 1461; XV, p. 485. Meyer, "Stahl und Eisen," 1900, p. 517. Münzel, "Stahl und Eisen," 1900, pp. 315, 382. Lürmann, "Stahl und Eisen," 1898, pp. 247, 361, 495, 806; 1899, p. 473; 1901, pp. 433, 489; 1902, pp. 898, 1009, 1067. Martens, "Stahl und Eisen," 1903, p. 735. Kirchhoff, "Journal Franklin Institute," 1903, CLVII, p. 81. Westgarth, "Iron Trade Review," August 3, 1903. Gruber, "Stahl und Eisen," 1904, pp. 9, 89. Wehrum, "Studies of Blast-furnace Gas and Its Most Economic Use," Scranton, Pennsylvania, 1900.

Blowing engines in this country[1] are mounted as: Horizontal direct with fly wheels (also compound); vertical direct with fly wheels (single, duplex, triplex, and beam); vertical beam engines.

(1) Horizontal Direct with Fly Wheels (also compound). Figures 67–68[2] show the general arrangement of a single and duplex engine: *a*, steam cylinder; *b*, air cylinder with flap valves; *d*, fly wheel; *f*, broad guides. All single engines have the great disadvantage that when the steam pressure is at a maximum the blast pressure is at minimum, and *vice versâ*. This is shown by the two diagrams,[3] Figures 69–70, representing indicator cards of steam cylinder (diameter 36 inches, stroke 48 inches, revolutions thirty-four, cut-off at one-quarter stroke, spring 40 pounds, steam pressure 40 pounds, throttle wide open, average effective pressure 30.6 pounds) and air cylinder (diameter 48 inches, stroke 48 inches, revolutions thirty-one, spring 10 pounds, blast pressure by gauge 4.5 pounds, average pressure 3.97 pounds). The difficulties are in part overcome by heavy fly wheels, and done away with by duplex engines with cranks at 90° (Figures 71–74). When the power of steam cylinder I is weakening, that of steam cylinder II is the strongest and does the increasing work of air cylinder I; in the same way steam cylinder I will supplement steam cylinder II when its power is decreasing. The advantages of horizontal engines are simplicity of construction, accessibility, and moderate foundations. The disadvantages are shortness of stroke, and wear of piston, piston rod, and cylinder on the lower side. They are not much in use in this country. The leading representative is the Bethlehem Steel Company, Bethlehem, Pennsylvania. Most of its engines are compounded. They have two high-pressure steam cylinders to one air cylinder. Further, the weight of the pistons is carried by steam pressure applied in chambers on the lower side, the steam being admitted through the hollow piston rods. Such an engine, *e.g.*, has a steam cylinder 30 and 54 inches in diameter, with an 80-inch stroke, the cut-off being at one-third; the air cylinder is 80 inches in diameter, capacity 400 cubic feet per revolution; it makes thirty-six revolutions per minute, and gives a pressure of 13–14 pounds.

(2) Vertical Direct with Fly Wheels (single, duplex, triplex, and beam). Figures 75–76[4] are sketches of two such engines, single and triple. *a* = steam cylinder, *b* = air cylinder, *c* = crosshead, *d* = fly wheels. Two piston rods are frequently used instead of one, as shown in figures. The single and duplex engine of this form is very common in this country. It is cheaply built, takes up little room, and is very accessible. There is a uniform wear of cylinder, pistons, and piston rods, and it permits a long stroke. It requires, however, very strong foundations; the crosshead is liable to break, and the wrist pins in the wheels tend to set up vibrations in them.

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[1] Ihreng, "Oesterreichisches Jahrbuch," 1894, XCII, p. 177.

[2] Beckert, *op. cit.*, p. 94.

[3] "Philadelphia Engineering Works Catalogue," p. 47, Figures 31 and 33.

[4] Beckert, *op. cit.*, pp. 95, 96.

BLAST–FURNACE BLOWING ENGINE OF AUTOMATIC CORLISS TYPE.

Diameter of steam cylinder.	Diameter of blowing cylinder.	Stroke.	Shop weights, pounds.	Revolutions per minute, ordinary speed.	Cubic feet of blast displacement per minute at ordinary speed.	Pressure of blast, designed to which these engines will work with good economy.
Inches.	Inches.	Inches.				
40	75	48	175,000	50	12,272	15 pounds per square inch.
40	84	60	230,000	40	15,392	12 pounds per square inch.
45	84	60	280,000	40	15,392	15 pounds per square inch.
45	90	60	290,000	40	17,700	12 pounds per square inch.

BLAST–FURNACE BLOWING ENGINES OF THE VARIABLE PUPPET VALVE CUT–OFF TYPE.

Diameter of steam cylinder.	Diameter of blowing cylinder.	Stroke.	Shop weights, approximate pounds.	Revolutions, ordinary speed.	Displacement of piston per minute, ordinary speed.	Maximum blast pressure for regular work.
Inches.	Inches.	Inches.			Cubic feet.	
28	66	36	80,000	60	8,550	10 pounds per square inch.
32	72	48	106,000	50	11,308	12 pounds per square inch.
36	84	60	165,000	40	15,392	11 pounds per square inch.
42	90	48	170,000	50	17,700	13 pounds per square inch.
48	96	60	280,000	40	20,000	15 pounds per square inch.

Illustrations: "Iron Age," December 18, 1890; February 19, 1891; February 26, 1891; September 17, 1891; September 26, 1897 (cross-compound). "Cassier's Magazine," XI, p. 124. "Iron Trade Review," March 5, 1903.

The vertical direct duplex engine with beam has the advantage over the ordinary duplex engine in that it gets along with a single fly wheel of comparatively small weight. Figures 77–78 represent a double engine. Such an engine at the works of the Maryland Steel Company, Sparrow's Point, Maryland,[1] has a steam cylinder 44 inches in diameter, air cylinder 84 inches

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[1] "Iron Age," May 21, 1891.

in diameter, stroke 60 inches; the cast-steel beam is 17½ feet long; the fly wheel 24 feet in diameter and weighs 36 tons.

(3) Vertical Beam Engine. This is shown in Figure 79. While these engines have most of the advantages of the vertical engines, they occupy much space, are not readily accessible, and are expensive to build.

The valves in all the diagrams are flap valves. In many cases these have been superseded by disk and puppet valves. The latest air cylinders have positively operated metal valves. The air valve is the weak point in blowing engines. The limit of speed that the air valves will work with must be that of the machine.[1]

It is the universal rule in this country that each furnace shall have its own engine or pair of engines, as only in this way is it possible to give a furnace its proper amount of air. Of two furnaces, one with a coarser, the other with a finer ore charge, that with the coarser will receive a greater volume of air. Further, in case of a breakdown all the furnaces would be affected instead of a single one. It was J. Kennedy who, in 1871, began to run a furnace on the single-engine principle. All engines have revolution counters, which show the amount of air a furnace is being worked with.

B. *The Equalizer.* The use of an equalizer is to reduce to a minimum the pulsations of a blowing engine, so that the furnace shall receive a constant stream of air. The weighted piston has been given up with blast furnaces, and is replaced by a long sheet-iron pipe of large diameter, which connects the blowing engine with the furnace.

C. *Pressure and Volume of the Blast.* The pressure of the blast needed depends upon the diameter of the hearth, the density of the fuel, and the size of the particles making up the charge. A large hearth requires a high pressure that the blast may penetrate to the centre. Dense (coke) fuel requires a higher pressure blast than when porous (charcoal). The pressure of a certain volume of air to enter the furnace will be greater if the charge is fine than if it is coarse.[2] The pressure of the blast causes the air to penetrate through the charge and to become evenly distributed. This causes the fuel to burn evenly and quickly, and thus localizes the heating. The result is a greater reduction of Fe_2O_3 by CO, and consequently a saving of fuel. The pressure is limited by the air currents meeting and interfering with one another. Charcoal blast furnaces are worked with 4–6 pounds; coke blast furnaces, 8–25, average 15, pounds; anthracite, 7–10 pounds pressure to the square inch.

A large volume of air will burn much fuel, hence produce much heat and put through many charges; but by increasing the velocity of the ascending gas current, it makes the top of the furnace hot and causes a waste of heat; on the other hand, it has the advantage that it works against the reduction of CO_2 by C and assists the reduction of Fe_2O_3 by CO. Data of volumes of air used in blast furnaces are given in the blast-furnace table. Of course, if pressure and volume are too great for a given furnace, the ore will pass too quickly through the CO-reduction zone of the furnace and arrive partly reduced only in the smelting zone, where the iron oxide has to be reduced by solid C, which cools the furnace. An iron ore passes through a furnace in twenty-four to sixty hours.

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[1] J. Kennedy, "Transactions American Institute of Mining Engineers," XXII, p. 537.

[2] Grammer, "Flue Dirt and Top Pressure in Iron Blast Furnaces," "Transactions American Institute of Mining Engineers," February, 1903.

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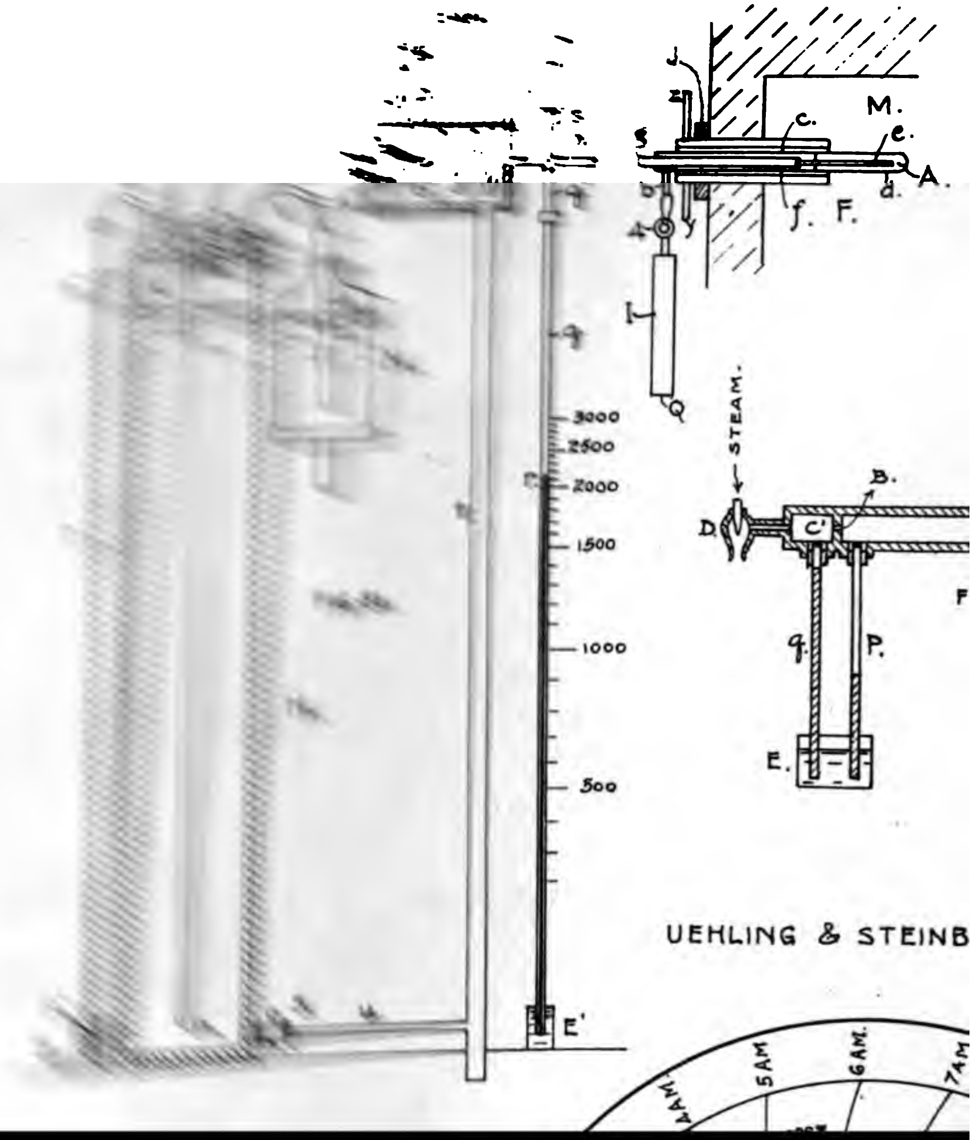
ASTOR, LENOX AND
TILDEN FOUNDATIONS.



UEHLING & STEINB

D. *Theory of Hot Blast.* All iron blast furnaces are now run with hot blast; the seeming disadvantages with charcoal iron blast furnaces, which for some time led to the retention of cold blast (cold-blast charcoal iron), disappeared with change of charge. The great advantage of hot blast is a saving of fuel. This is shown by the following figures of Bell.[1] The causes of this are: (1) the sensible heat brought to the furnace by the hot blast; (2) the smaller amount of air required to burn the reduced percentage of fuel that is necessary — the result is that a smaller volume of gas is produced, which of course gives off its heat more readily to the descending charges than would the larger volume with cold blast, the gases, therefore, passing off at the throat with a lower temperature; (3) the more energetic action of hot blast with more complete combustion of the C to CO, and a resulting localization of the heat; (4) a greater smelting power in consequence of the more energetic combustion; (5) an increased capacity for ore on account of the smaller amount of fuel charged. Beside a practical there is a theoretical limit to the heating of the blast. The carbon of the fuel has two functions — generation of heat, and furnishing of CO for the reduction of Fe_2O_3. With blast that is excessively superheated (1) there will be lack of CO on account of the diminished amount of fuel charged; the Fe_2O_3, imperfectly reduced by CO, will have to be reduced by solid C, which is wasteful, and counteracts the advantage of superheating. (2) If the throat of the furnace is too cool, on account of too small a volume of ascending gases, the ores arrive insufficiently prepared in the upper reducing zone, and CO cannot have its full effect upon them. (3) If the furnace is low the ores will not be exposed a sufficient time to the action of CO. (4) The limit is reached earlier with readily reducible ores than with ores difficult to deoxidize, as even with cold blast they require less fuel; with the production of white iron low in Mn and Si earlier than with irons rich in these elements; and lastly, with a porous fuel (charcoal) earlier than with one that is dense (coke, anthracite). Bell[2] concludes that it is not economical to raise the temperature of the blast beyond 930° C. In practice large coke or anthracite furnaces producing iron high in Mn and Si have blast temperatures reaching 800° C.; when making iron low in Mn and Si the temperature limit is at about 500° C.; charcoal furnaces, of smaller capacity than coke furnaces, have blast temperatures of 400–500° C. when making gray iron, and of 300° C. when making white iron.

Height of furnace.	Blast.	Coke per 2,000 pounds of iron.
42 feet	Cold	6,000 pounds
48 feet	Hot	2,892 pounds
70 feet	Cold	3,800 pounds
80 feet	Hot	1,999 pounds

E. *Measurement of Blast and Gas Temperature.* In a modern blast-furnace plant a continuous record is kept of the temperature of the blast, as well as that of the escaping furnace

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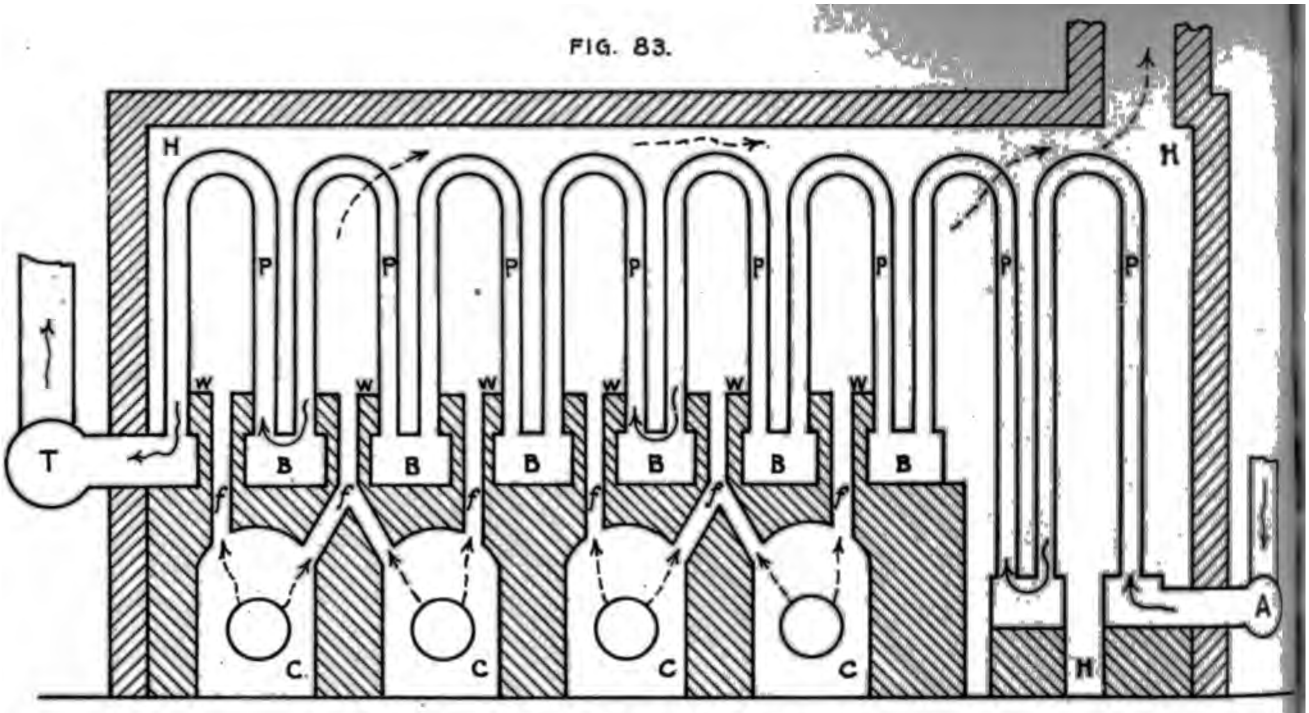
[1] "Journal Iron and Steel Institute," 1893, II, p. 221.

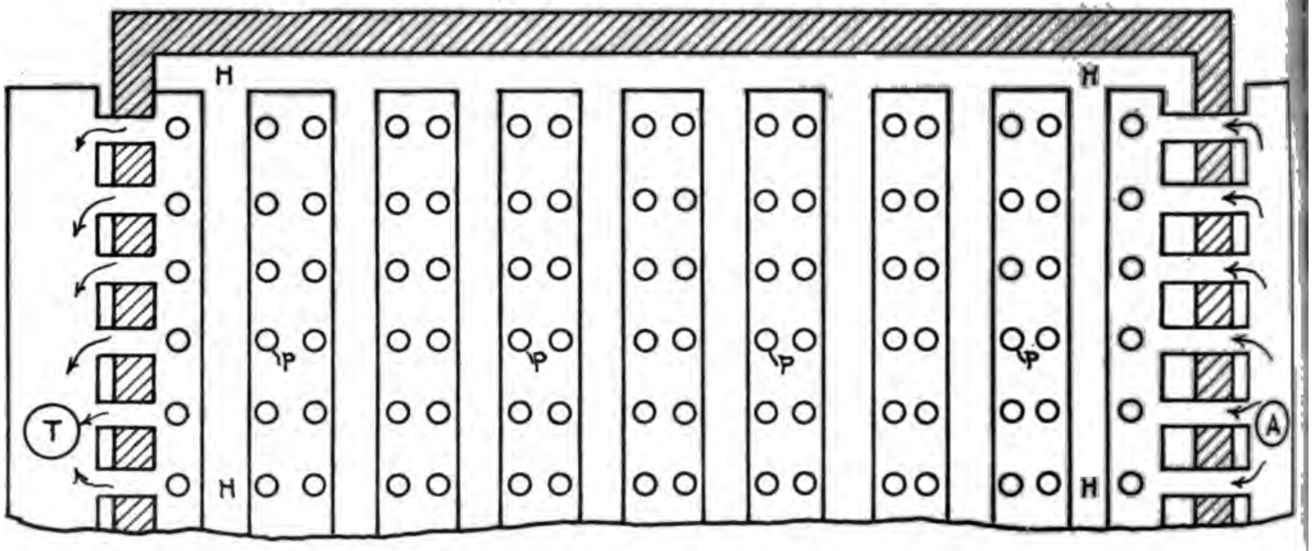
[2] "Principles," p. 92.

FIG. 83.



LONGITUDINAL VERTICAL SECTION.

FIG 84.

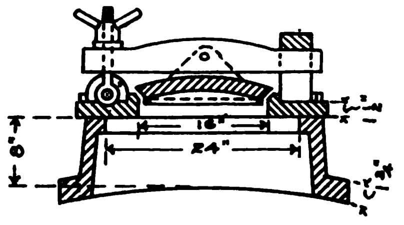


A- COLD AIR DISTRIBUTING PIPE
B- BED-PIPE
C- COMBUSTION CHAMBERS
f- FLUES LEADING HOT-AIR AND FLAMES TO THE PIPES, P.P.

HORIZONTAL SECTION
— PIPE - STOVE —

P- OX-BOW IRON PIPE
W- BRICK WALL.
T- HOT-AIR COLLECTING PIPE
H- BRICK CHAMBER.

FIG. 98.



AIR-INLET VALVE.

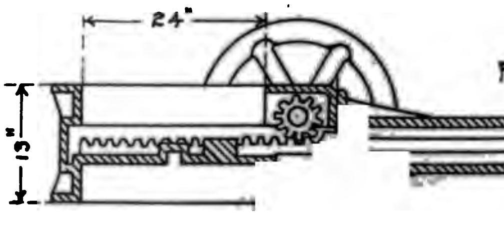


FIG.99.

COLD B

gases. Of the many pyrometers used for the purpose that of Uehling and Steinbart (Newark, New Jersey) has come into almost universal use. It is based on the viscosity of air when flowing through fine orifices. (Figures 80–81.) Supposing (Figure 80) air to be sucked into the chamber *C*, through the orifice *A*, and out of it through a second orifice *B*, into the chamber *C'*, and thence discharged through *D*, the tension in the two chambers will be a function of the temperature. If the orifice *A* be connected with the hot-blast pipe and the air cooled down to a fixed temperature before it passes through *B*, less air will be sucked in at *A* than is drawn off through *B*, giving a different tension in chambers *C* and *C'*, which can be measured by the arms *p* and *q* of the manometer. If the suction is a constant and the temperature at *B* is fixed, any change in the temperature of the hot blast will cause a corresponding change of tension in the chamber to be read off on the manometer. Figure 81 represents diagrammatically the different parts of the instrument. Into the chamber *M*, the temperature of which is to be measured, reaches the fire tube *F*. This consists of a platinum tube, *d*, enclosing a smaller one, *e*, having the small orifice *A*. Both tubes are brazed into copper drawn tubes, *c* and *f*, surrounded by a water jacket with in- and out-flows, *y* and *z*. The air, sucked in by the aspirator 6, enters, through the filter *l*, a pipe filled with cotton, to be purified, travels through the regulating cock 4 and pipe *b* into the annular space between *f* and *c* and their continuations, *d* and *e*. Here it becomes superheated and enters at *A* the pipe *e*, which forms one end of the chamber *C* (Figure 80), and travels through *fgh* (with branches *q'* and *S* leading to the manometer *q* and the recording gauge *L*) and the coil *i* to the opposite end at *B*. This end is kept at the constant temperature of 100° C. by the aspirator *D* exhausting into the chamber *G*, from which steam and water escape through the pipe *t* at atmospheric pressure. In order to insure constant suction, the aspirator 6 draws the air through pipe *m*, let into the top of a Mariotte flask, *C'* (with tube, *nn'*); into it the air from *i* discharges through cock 2. The suction is regulated by cock 1 so that the water in the manometer *r* shall be at the level of the pointer *r'*. Any evaporated water is replenished from the flask *N* by opening the cock 3.

Figure 82[1] is an automatic record of the blast and gas temperatures of a blast furnace from two pyrometers.

F. *Method of Heating Blast.* In 1828 Neilson[2] superheated air to aid combustion by passing it through an iron vessel suspended over a fireplace; air thus heated to 100° C. proved successful in the saving of fuel at a blast furnace of the Clyde Iron Works, Glasgow, in 1829. The foundation of the modern superheated blast was laid by Faber du Faur at Wasseralfingen in 1837, who withdrew the gases from beneath the throat of the blast furnace and used them for superheating the blast instead of burning extraneous fuel.

According to the material employed for heat transference there are two classes of stoves for heating blast — pipe stoves and brick stoves.

G. *Pipe Stoves* permit heating the blast up to 500° C., and are continuous in their action. The cold air from the blast main enters a branching battery of U-shaped pipes at one end of a heated brick chamber, and after passing through the pipes the hot blast is gathered into the hot-blast main at the other end and conveyed to the bustle pipe and the tuyères of the blast furnace. Figures 83–84 represent a sketch of pipe stoves commonly used in the Lehigh Valley.

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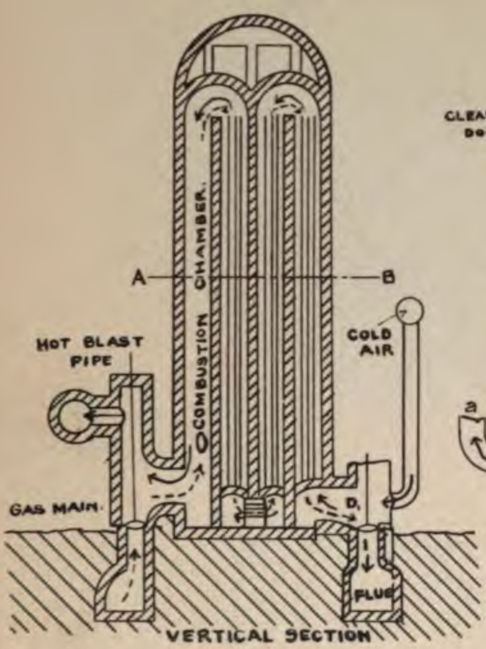
[1] "Iron Age," October 29, 1896, p. 814.

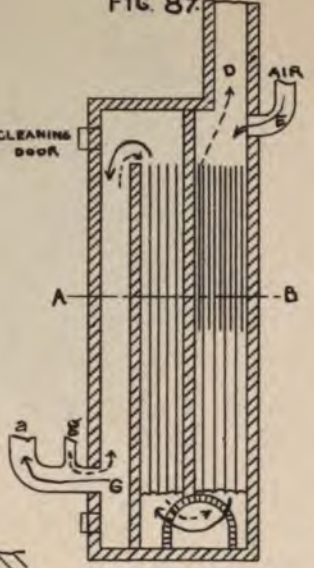
[2] "Stahl und Eisen," 1895, p. 509.

FIG. 85.



VERTICAL SECTION

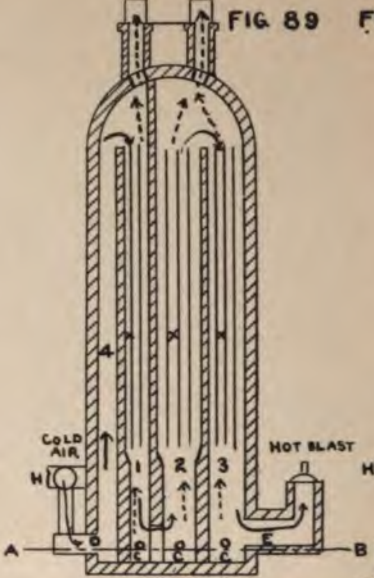
FIG. 87.



VERTICAL SECTION

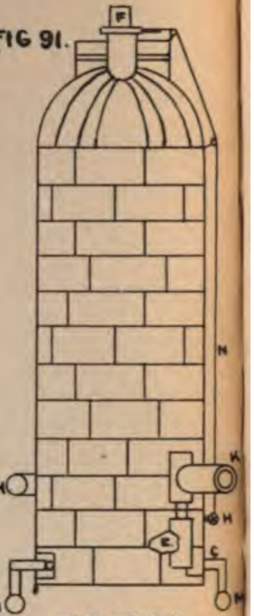
GORDON THREE PASS STOVE

FIG. 89



VERTICAL SECTION.

FIG. 91.

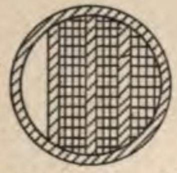


ELEVATION.

HUGH KENNEDY STOVE.

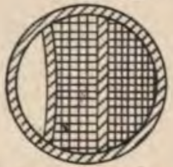
FIG. 86

WHITWELL FOUR PASS STOVE.



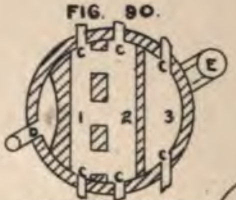
SECTION ON AB.

FIG. 88.



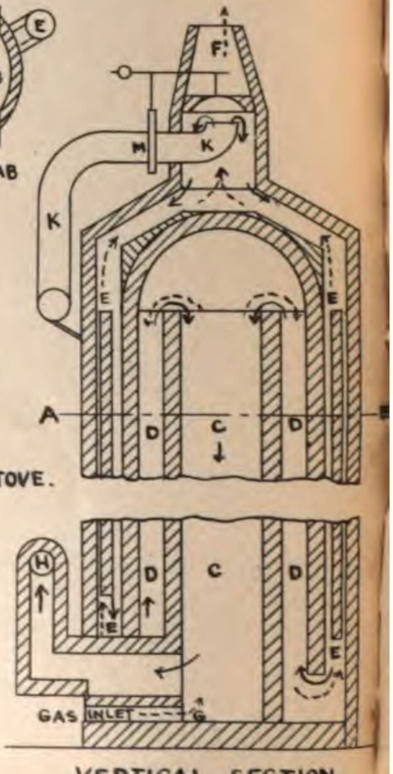
SECTION ON AB.

FIG. 90.



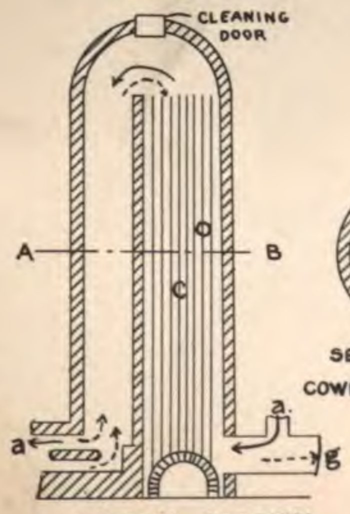
SECTION ON AB

FIG. 92.



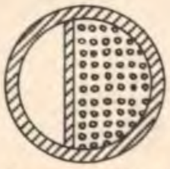
VERTICAL SECTION.

FIG. 94.



VERTICAL SECTION.

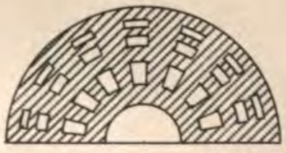
FIG. 95.



SECTION ON AB

COWPER-KENNEDY STOVE

FIG. 93



HALF-SECTION ON AB.

MASSICKS & CROOKS' STOVE.

Legend: a = cold-air pipe; b = bed pipe; c = condensation chamber; f = flues leading hot gases and flames to the U-shaped (oxbow) cast-iron pipes p; w = brick wall to protect p against excessive heat; T = hot-air collecting pipe; F = flue leading off products of combustion which travel in a direction opposite to that of the air. It is not advisable to make pipe stoves very large, as the pipes are liable to be heated unevenly, but to increase the number of stoves. The heat of the burning gas is utilized the better the larger the heating surface of the pipes and the slower the speed of the air. In order to raise the temperature of 1 cubic foot of air per minute from say 15° C. to 450° C. not less than 0.6 square foot of heating surface will be required, supposing the velocity of the air current not to exceed 50 feet per second. Ledebur[1] says that the iron best suited for pipes should not contain over 0.5 P, 3–3.5 C, 0.7–1.0 Si, 0.5 Mn. After being in use for some time pipes become oxidized on the inside and do not transfer the heat so well as at first; they also are liable to become leaky. Besides the apparatus shown in figure with standing pipes, there are others with suspended and with horizontal pipes. Pipe stoves are still used with charcoal and anthracite furnaces, but even there they are giving way to

H. *Brick Stoves*, first introduced by Cowper in 1860.[2] They heat the blast up to 800° C. Hartman[3] states that in practice it has been found that by replacing pipe stoves heated to 480° C. with brick stoves heated to 760° C. a saving of 15 per cent. of fuel and an increase of 20 per cent. in the make of iron was effected. Stoves are cylinders of boiler iron with a dome-shaped top, filled with fire brick of different shapes and forms having passages through which gases or air may pass freely. Between the iron shell and the brickwork there is an expansion space (similar to 75 in Figure 11) 1½–2 inches wide, and filled with loose material. Brick stoves are mounted in groups of three or four for a furnace, and are intermittent in their action (regenerative); *i. e.*, there are two periods in operating them, one of storing up heat and one of giving out heat. When two or three stoves are being heated up by burning gas in them (they are on gas) one is giving out the stored heat to the cold blast passing through it (is on air); gas and air travel alternately in opposite directions. From 5 to 6 square feet of heating surface are necessary for 1 cubic foot of air per minute, the velocity of the cold air being 6–6½ feet, that of the hot air 16–26 feet per second. Stoves are 18–24 feet in diameter and about 60 feet high; in the last few years the height has been increased to 80 and even 100 feet. The fire brick must be of a very refractory character, as should they become glazed by heat alone or by the fuel dust (gas ashes), which is rich in iron, they would practically cease to absorb any heat. The construction of a stove must be such as to permit a ready removal of these ashes, which are always carried along by the gases to a greater or less extent.

All brick stoves may be considered as belonging to one of two types: the Whitwell, in which gas and air make more than two passes; and the Cowper, in which they make two passes only.

(1) The Whitwell Stove.[4] The Whitwell stove has undergone many changes in the details of construction since the first was erected in 1869. Figures 85 and 86 represent a four-pass stove (diameter, 18 feet; height, 60 feet; heating surface, 29,000 square feet), gas or air

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[1] "Eisenhüttenkunde," p. 490.

[2] General Review, Lürmann, "Stahl und Eisen," 1890, p. 766; 1892, p. 568; 1901, p. 785. Teichgräber, *op. cit.*, 1902, p. 323. Thwaite, "Journal Iron and Steel Institute," 1903, I, p. 246.

[3] "Journal Franklin Institute," January 22, 1886.

[4] "Transactions American Institute of Mining Engineers," IX, p. 480.

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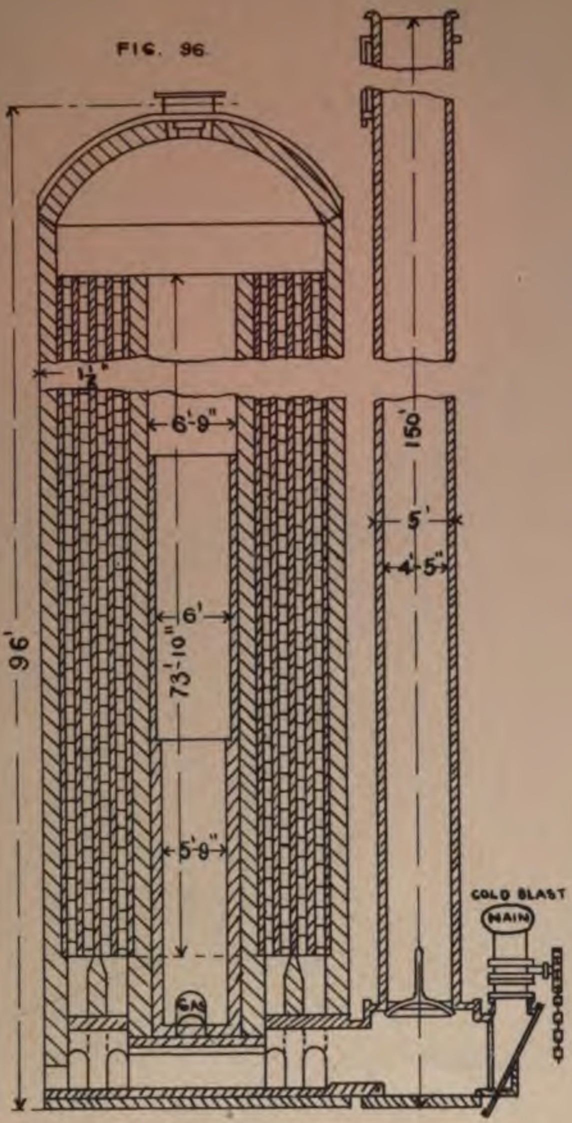
FIG. 96.



VERTICAL SECTION.
COWPER KENNEDY STOVE WITH
CENTRAL COMBUSTION CHAMBER.

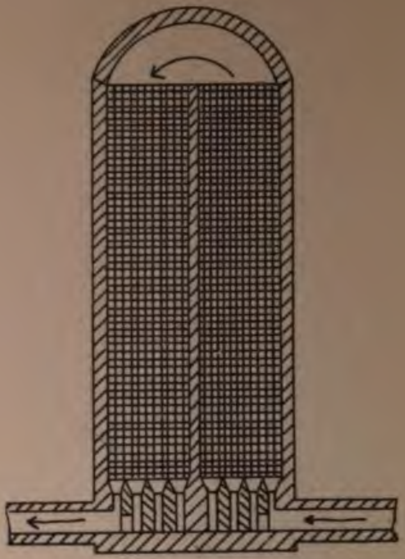
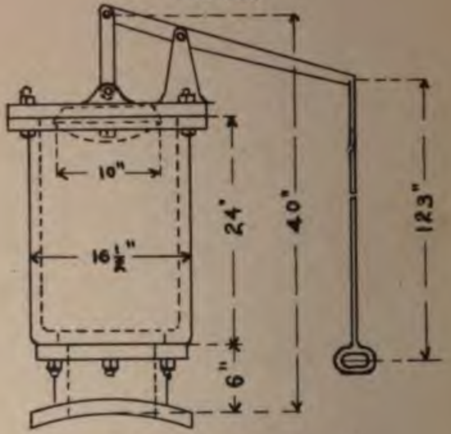


FIG. 102.
GJERS HOT-BLAST EQUALIZER

FIG 101



GAS ESCAPE VALVE

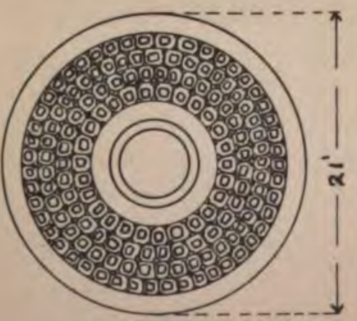
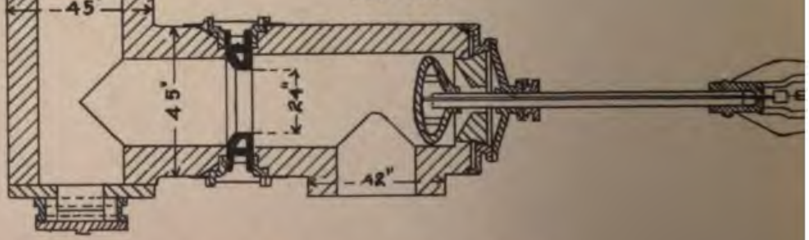


FIG. 97.
FORM OF CHECKER TILE.

FIG. 100



HOT BLAST POPPET VALVE

making four passes before it enters the chimney flue or the hot-blast pipe. To the left is the combustion chamber. The gas coming from the main ascends in the chamber, meets the air drawn in through the air port, and burns. Any imperfect combustion in the chamber is finished in the dome. The products of combustion zigzag through the checkerwork (oblong flues) in the three regenerative chambers, as shown by the dotted arrows. These chambers rest on brick arches, leaving open spaces beneath them. The gas while burning becomes heated, expands, and ascends; after the combustion it does not expand any more; on the contrary it contracts while heat is being absorbed, and at the same time descends in the first flue. The combustion chamber, therefore, has no checkerwork; that of the second and third pass has large flues, that of the fourth small flues. Instead of all the gas being burned in the combustion chamber, sometimes only part of the air required is admitted to this chamber and the rest at the top or bottom of the other three chambers through special air ports. This is called burning back, and is accompanied by an increase in temperature of the blast, owing to the better combustion of the gas to CO_2. The cold blast going in the opposite direction, as shown by the full-drawn arrows, first ascends through the smaller flues, then taking up more and more heat expands, zigzags through the two larger flues, and finally descends in the largest flue, the combustion chamber, and passes out through the hot-blast pipe to the blast furnace. In the figure the two water-cooled puppet valves and sliding valve are set for the passage of the cold blast.

(2) The Gordon Three-pass Stove.[1] Figures 87 and 88 represent a stove 23 feet in diameter and 60 feet in height. Gas enters at *g* and, after three passes, leaves the stove through a chimney, *D*, at the top; air enters at *E* and leaves at *a*. The valves have been omitted in the sketch. In order to secure the same heating surface as in the four-pass Whitwell stove, the number of the flues in the upper part of the pass under the chimney has been increased and the thickness of the brickwork correspondingly decreased, thus making the heating surface the same as in the Whitwell stove. Each stove has an individual chimney, which does away with the expensive underground flues and large central stack required by a set of Whitwell stoves.

The Hugh Kennedy Stove.[2] Figures 89–91 represent a Kennedy stove 20 feet in diameter, 60 feet in height. The gas from the main *M*, instead of burning in a combustion chamber and then making four (Whitwell) or three (Gordon) or two (Cowper) passes, burns in three chambers and makes only one pass, entering near the bottom through burners in the openings, *C*, and leaving at the top through small stacks, *F*, provided with dampers, *L*. The cold air entering at *H* makes its four passes as in the Whitwell stove, leaving at *K* as hot blast.

(3) The Massick and Crooke Stove.[3] This is a three-pass stove. Figures 92–93 represent a stove 18 feet in diameter and 60 feet in height. The gas entering at *G* passes into the central combustion chamber and burns. The products of combustion descend at the top in the annular flues, *D*, and then ascending through the flues of a double annular chamber, *EE*, pass out through the stack *F* at the top, the valve *N* having been lowered. The cold blast, passing by the slide valve *M*, enters the stove at the top, valve *N* having been closed, and, traveling in a direction opposite to that of the gases, leaves the stove through the hot-blast pipe *H*.

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[1] "Transactions American Institute of Mining Engineers," XIV, p. 159.

[2] *Op. cit.*, XXI, p. 720.

[3] *Op. cit.*, XIX, p. 1036.

Three Stoves.		Weight of Iron Parts.		Weight of Brick.		Cost.
Diameter.	Height.	Wrought iron.	Cast iron.	First class.	Second class.	
16 feet 6 inches	60 feet	175,000 pounds	41,000 pounds	186,000 pounds	203,000 pounds	$17,521
18 feet	60 feet	188,500 pounds	42,000 pounds	235,000 pounds	224,000 pounds	19,934
19 feet 6 inches	65 feet	220,000 pounds	46,000 pounds	256,000 pounds	342,000 pounds	23,839

(4) The Cowper-Kennedy Stove.[1] It is an ascertained fact[2] that in the Whitwell stove 70 per cent. of the heat of the gas is taken up by the combustion chamber and the first downcomer. If the heating surface of the downcomer be sufficiently large, one down pass should be enough to absorb practically all the heat. This is the ground idea of the Cowper stoves, all of which have two passes. Figures 94–95 represent a 21-foot x 72-foot 6-inch Cowper-Kennedy stove. The bricks forming the flues are hexagonal; they were 10 inches across and had circular openings 6 inches in diameter. These dimensions[3] have been changed to 12 inches across and 9 inches diameter. The bricks are 12 inches deep, except in the bottom and top courses, which are 6 inches deep.

(5) The Cowper-Kennedy Stoves with Central Combustion Chamber.[4] Figures 96–97 represent the stoves of the Duquesne blast-furnace plant. They are 27 feet in diameter and 97 feet high. The tiles have openings approximately 9 inches square, with filleted corners. Each stove has an independent chimney 130 feet high and 5 feet in diameter.

(6) The Ford and Moncur Stove[5] is an English stove similar to the above Cowper-Kennedy, with central combustion chamber. Other stoves of this type are those of Cabot-Patterson[6] and Moore.[7] This type of furnace is growing in favor.

(7) The bricks of the different stoves show a great variety of forms[8] and ways of joining so as to give large and smooth cooling surfaces and to allow for expansion and contraction.

(8) The valves found in a stove are the air valves, cold- and hot-blast valves, and gas-escape valves. Figure 98[9] represents an air-inlet valve of cast iron, with heavy flanges; the spherical valve is firmly held in the beveled seat by a cast-iron bar secured to the frame by a fixed leg and a link bolt. Cleaning doors are of the same general construction. The cold-blast valve shown in Figure 99[10] is of the gate type and is operated by means of rack and

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[1] "Transactions American Institute of Mining Engineers," X, p. 495.

[2] Coffin, *op. cit.*, XXI, p. 720.

[3] "Iron Age," July 10, 1890.

[4] *Op. cit.*, March 25, 1897; August 6, 1903.

[5] "Journal Iron and Steel Institute," 1896, I, p. 20.

[6] "Iron Age," October 10, 1901.

[7] *Op. cit.*, February 12, 1903.

[8] Lürmann, "Stahl und Eisen," 1884, p. 484; 1890, p. 766; 1896, p. 907. Roberts, "Iron Age," January 9, 1896. Kennedy, *op. cit.*, January 10, 1890.

[9] "Philadelphia Engineering Works Catalogue," p. 102.

[10] *Op. cit.*, p. 97.

pinion. Hot-blast valves are either sliding or puppet valves. The sliding valves are similar to the cold-blast valves, only that the gate and the seat are air-cooled, being cast around a course of 1-inch pipe. The puppet valves (Figure 100)[1] are water-cooled. The seats are made of bronze and are hollow, and air- or water-cooled. The valve stem is of seamless drawn tubing. The gas-escape valve (Figure 101)[2] for the hot-blast main serves to carry off the gas which backs up from the furnaces when the blast is shut off. The valve, being heavier than the lever, is held in position by the pressure of the blast; when this is taken off it falls, and closes up again when the blast is put on.

(9) In Operating a Stove the gas is admitted through the gas valve. After it has burned three to four hours, the gas and chimney valves, the burner and the air-inlet openings are closed, the cold-blast valve and then the hot-blast valve are opened. The stove now remains one and one-half to two hours on air. The next stove is then cut out from the blast furnace (the hot-blast, then the cold-blast valve is closed and the chimney valve opened) and the stove put on gas, *i.e.*, the air- and gas-inlet valves are opened. The temperature of the blast should not vary over 10° C. between the changes. In order to do away with all variations of temperature of hot blast Gjers[3] constructed a

(10) Hot Blast Equalizer for the Ayresome Iron Works, Middlesbrough, England. It is (Figure 102) a brick stove with checkerwork having a vertical partition wall passing through the centre. The hot blast from a freshly heated stove passing up one-half the checkerwork and down the other gives up some of its excess of heat to the checkerwork before it passes on to the furnace. When the temperature of the hot blast tends to sink below that of the checkerwork of the equalizer it takes up the excess of heat it had given up before and leaves the equalizer at a normal temperature. The equalizer thus forms a large reservoir for heat, correcting any variation of temperature of the hot blast.

At certain intervals the flues in brickwork have to be freed from

(11) Accumulated Dust. This is done by lowering with a chain a weighted disk which will scrape the sides clean. Thwaite[4] studied the effect of flue dust upon the thermal efficiency of hot-blast stoves.

SECTION 12. **Pumps.** Around a blast furnace very much water is needed for cooling the furnace walls, for the boilers, condensers, etc., and pumps of large capacity are required. They are usually placed in the engine house with the service tank (an elevated water tower) outside. Water connection is often made with the feed water of the boilers (run at from 80 to 140 pounds pressure) so as to permit flushing water blocks and other coolers should they become clogged or choked up. A furnace making 350–400 tons pig iron in twenty-four hours requires 3,000,000 to 4,000,000 gallons water for cooling with a head of 30 pounds at furnace level.

SECTION 13. **Boilers.** In old plants there are still found elephant boilers; these have been replaced by two-flue boilers, and the latter have had to give way to Heine Safety, Babcock and Willcox, Cahall, and other modern boilers. The boilers are all fitted for burning coal as

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[1] "Philadelphia Engineering Works Catalogue," p. 92.

[2] *Op. cit.*, p. 80.

[3] "Engineering and Mining Journal," September 16, 1899; "Journal Iron and Steel Institute," 1900, I, p. 154; 1902, II, p. 282; "Iron Age," August 27, 1903; May 19, 1904.

[4] "Journal Iron and Steel Institute," 1903, I, p. 246.

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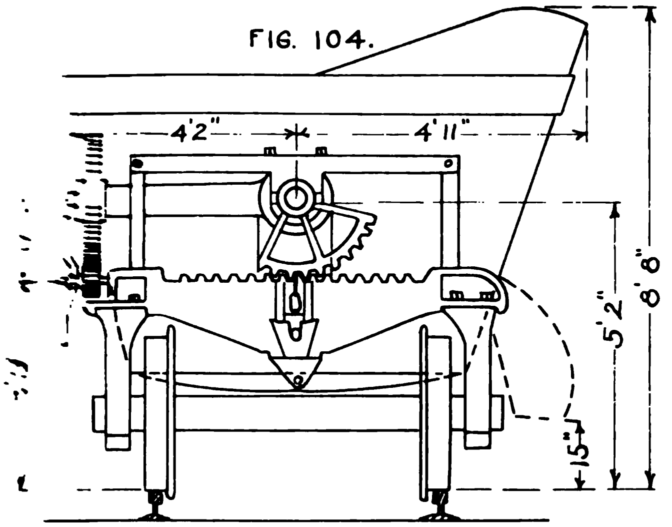
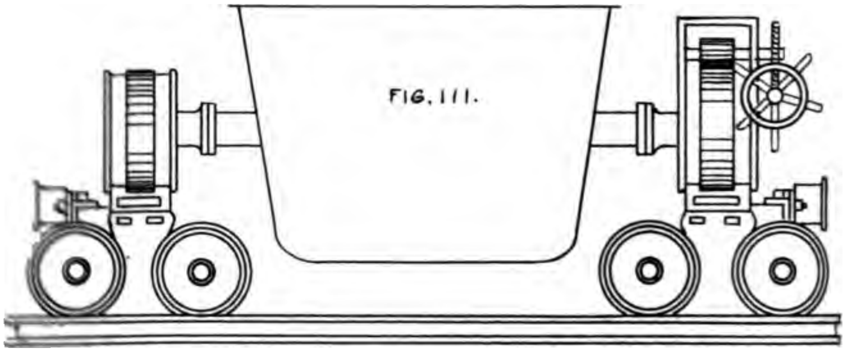
FIG. 104.



HOT METAL LADLE.

FIG. 111.



THE WEIMER GONDOLA CINDER CAR.

FIG. 112.

RAMSAY SLAG POT CLEANING DEVICE.

FIG. 113.

well as gas. They usually are run on gas, coal being used only when something is wrong with the furnace, which may either not yield enough gas or give it off in irregular amounts. A 300-ton furnace requires a 2,500-horse power boiler plant. A boiler plant for a blast furnace costs erected about $15 a horse power, excluding buildings. A Cahall boiler of 250 horse power costs about $2,400, and the erection $1,000 to $1,500.

SECTION 14. **The Chimney.** If a plant has a large central stack for the stoves, it usually also does duty for the boilers. Stacks are generally round, of plate iron, and lined up with fire brick. Their height is not less than 70 feet and sometimes 235 feet and more.

SECTION 15. **The Stock House.** This is usually placed just behind the blast furnace. It is often covered to protect the ore, flux, and fuel from the weather. When large quantities of ore have to be stored, the ore piles remain uncovered. It is not often that flux and fuel have to be stored, so they are more frequently under cover than in the open. The stock house is open at both ends, and has its tracks on a trestle running lengthwise.[1] The cars are usually bottom-discharge cars, and the raw materials are, if possible, collected in bunkers, from which they are drawn into buggies, four-wheel cars, or two-wheel barrows. Where bunkers are not used, the raw materials have to be shoveled into the buggies, which increases the expense.

SECTION 16. **The Hoist.**[2] In the hand-barrow system the buggies running on tracks bring the raw materials to the scales (of multiple-beam type) placed near the hoist tower. They are raised to the height of the top of the furnace, and then wheeled by the top-fillers over the bridge to it. A common form of steam hoist tower is shown in Figure 103.[3] It is of structural steel and double, say 12 x 23 feet, loaded buggies going up while the empties come down. The distance between centre of hoist and blast furnace is 35–50 feet. Hydraulic lifts have fallen into disuse; occasionally pneumatic hoists[4] are still in operation; electric hoists[5] are beginning to replace steam hoists. With mechanical top-charging,[6] as at the Duquesne, Eliza, and other furnaces, the filling of hopper and lowering of bells is effected from the ground floor.

SECTION 17. **The Casting House.** This is either a brick or an iron structure in front of the furnace, 80–150 feet long. Its roof extends back of the furnace. The floor (inclined about 2 feet) or the pig bed is filled either with common sand or is composed of heavy cast-iron molds (chills) when it is desired to have pigs free from adhering sand. The sand bed is molded up before each cast (the pigs are 4 and $3\frac{1}{2}$ inches wide, $3\frac{1}{2}$ feet long, and weigh 100 pounds); with chills the bed is ready when the pigs from the previous cast have been removed. One ton sand-cast pig iron = 2,268 pounds = 2,240 pounds pig iron + 28 pounds sand.

With direct-metal plants, *i. e.*, plants from which the molten cast iron goes to mixers and thence to the Bessemer or the open-hearth department or the casting machine on Sundays, the iron is run into ladles like the one of the National Steel Company, Youngstown, Ohio, shown in Figure 104.[7] They are of boiler iron and are lined on the bottom with a course of $2\frac{1}{2}$-inch

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[1] See plan of single-furnace plant, Figure 30.

[2] Hauer, "Hüttenwesenmaschinen," Leipsic, 1876; Supplement, 1887.

[3] "Philadelphia Engineering Works Catalogue," p. 8.

[4] "Transactions American Institute of Mining Engineers," XXVII, pp. 25, 26.

[5] "Stahl und Eisen," 1901, p. 1353.

[6] Treadwell Ore-Charging Scale Car, "Iron Age," December 11, 1900.

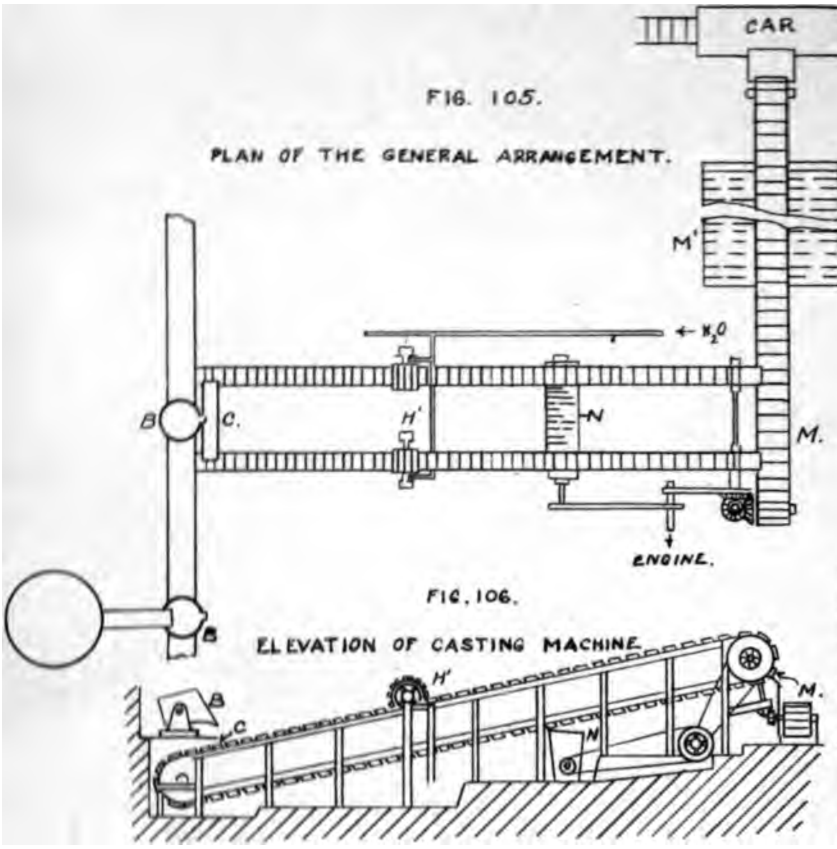
[7] "Iron Age," December 21, 1899; "Transactions American Institute of Mining Engineers," XVII, p. 20. A common form is represented in "Transactions American Institute of Mining Engineers," XXVII, p. 20; Treadwell ladle, "Iron Age," July 10, 1902; Pollock ladle, "Iron Age," August 14, 1902, or "Engineering and Mining Journal," August 16, 1902; Weimer ladle, "Iron Trade Review," April 23, 1903.

FIG. 105.

PLAN OF THE GENERAL ARRANGEMENT.

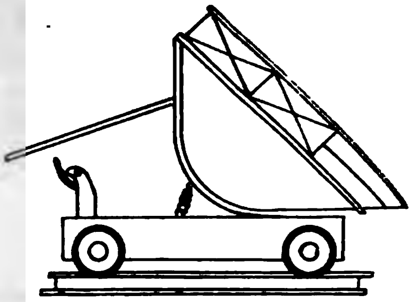
FIG. 106.

ELEVATION OF CASTING MACHINE.



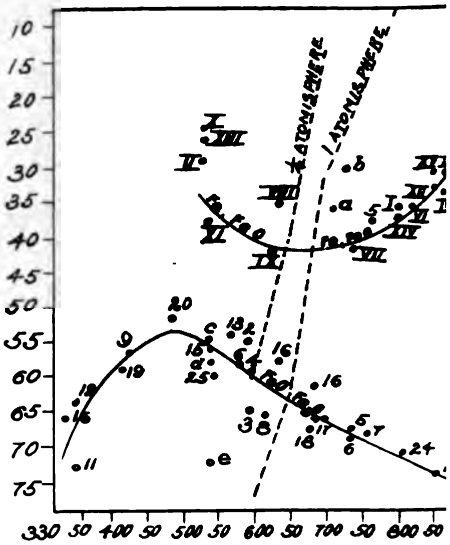
THE VEHLING PIG IRON CASTING MACHINE.

FIG. 110.

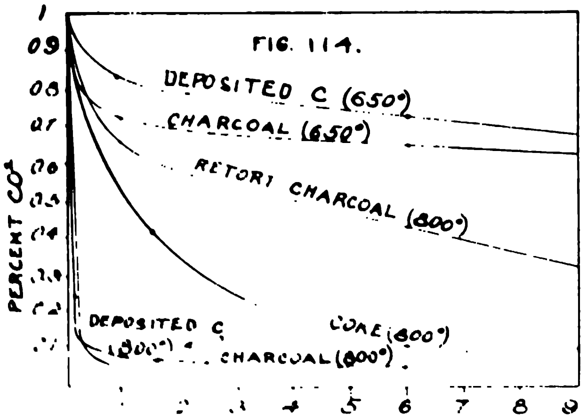


TAWS & HARTMAN CINDER CAR

FIG. 116.

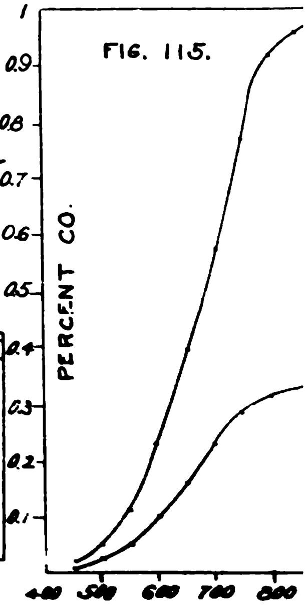


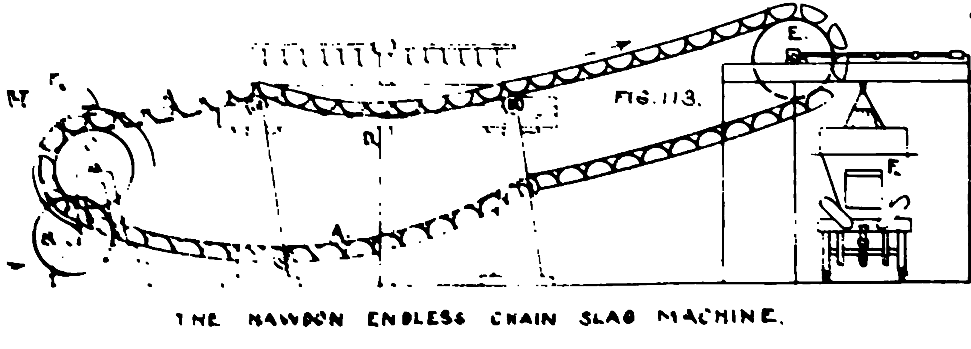
EQUILIBRIUM CURVES OF CO



REDUCTION OF -CO2- BY C.

SPLITTING OF-CO- INTO CO2 AND C.





THE HAWDON ENDLESS CHAIN SLAG MACHINE.

fire brick, then rammed with ganister and clay, and hold 15–18 tons of metal. They are dried and warmed for twelve hours by a jet of producer or natural gas. Hot Bessemer pig, after remaining six hours in the ladle, will still pour well, while spiegeleisen, chilling much more easily, requires a very heavy lining if it is to stand for any length of time.

SECTION 18. **Pig-Casting Machines.**[1] In order to reduce the handling of pig metal when cast into chills, pig-casting machines are being introduced into all modern plants.

The Uehling Casting Machine (Figures 105–106)[2] is substantially a link belt conveyor carrying a series of molds, D, which overlap, and thus permit continuous casting from a ladle, B, without making any scrap. A mold, cast in chills directly from the furnace, holds 120 pounds of iron and lasts from twenty-five to thirty days. C is a trough serving two strings of molds. In order to shorten the length of the conveyor, the pigs are sprinkled with water by means of the water distributer H' (a sprocket wheel with troughs, the water fed into them being delivered to the centre of the molds). A normal length is 85 feet between centres and the speed 15 feet per minute. The mold, arriving at the head of the conveyor, discharges the pigs into a conveyor, M, which, after passing through the cooling tank M', delivers them over an inclined chute, M_2, into the cars. The molds are whitewashed on their return to the casting end by means of jets through which milk of lime, kept in motion by a stirrer, N, is forced by an air-jet pump. A machine with two strings of molds, 85 feet long, running at the rate of 15 feet per minute, requires 15 horse power, and its capacity is 90 tons per hour. At the Lucy furnaces, Pittsburg, the machine handles on the average 750 tons in twenty-four hours. The cost per ton of handling pig iron in sand casting was, in one case, 11.2 cents; in machine casting, 7.2 cents; in another, 10 and 6.5 cents.

With small plants the pig-casting machine can be used to handle the slag. The Heyl and Patterson (Pittsburg, Pennsylvania) pig-casting machine[3] used by the Cambria Steel Company, Johnstown, Pennsylvania, and other works resembles very much that of Uehling. It has expensive pressed-steel molds, requires less space (as the filled molds are immersed in water), but more repairs; the molds are smoked instead of being whitewashed. Other American machines are those of Baker,[4] Davis,[5] Suppes,[6] Hartman,[7] Ramsay.[8] English pig-casting machines are those of Howdon-Wainford,[9] Bell.[10] German machines are described in "Stahl und Eisen," 1900, p. 1033; 1901, pp. 163, 852.

SECTION 19. **Disposal of Slag.** The slag was formerly run into large molds of sand (cinder wharf) near the furnace, and removed by a crane when cold or broken by hand and shoveled into cars. As a rule it is tapped into cinder cars,[11] hauled to the dump, and there poured; in a few instances slag-casting machines are in use.

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[1] "General Article," by Fay, "Iron Age," October 10, 17, 1901; "Machine- and Sand-cast Pig," by Colby, "Iron Age," June 20, 1901; "Foundry," 1901–02, XIX, p. 110; "Cassier's Magazine," 1903, XXIV, p. 134.

[2] "Iron Age," April 22, 1897; March 3, 1898; "Transactions American Institute of Mining Engineers," XXVII, p. 22; "Stahl und Eisen," 1897, p. 665; "Cassier's Magazine," 1903, XXIV, p. 13.

[3] Grammer, "Transactions American Institute of Mining Engineers," February, 1904.

[4] "Iron Age," May 19, 1898.

[5] "American Manufacturer," June 13, 1898.

[6] "Iron Trade Review," July 7, 1898.

[7] "Iron Age," September 22, 1898.

[8] "Iron Trade Review," December 13, 1900.

[9] "Journal Iron and Steel Institute," 1899, II, p. 52.

[10] "Iron Trade Review," August 8, 1901.

[11] "Stahl und Eisen," 1891, p. 370.

(1) *The Philadelphia Engineering Works' Cinder Car.* Figures 107–109[1] show a four-wheel truck with frame of 9-inch I-beams. Upon these roll three rockers carrying a bottom-plate cast in two sections. The plate is held level by pins, except when dumping; it often has a rim within which are placed brick to protect the bottom. The tub, the sides of which consist of six castings, has a capacity of 120 cubic feet and holds about 6 tons of cinder. The tub can be tipped endwise or sidewise, according as it is set. When the cinder is to be run off in the fluid state, the tap hole in the side is opened and only the shell dumped after the tub has been raised. A car similar to the one described is that of Bock-Koltz.[2]

(2) *The Taws and Hartman Cinder Car*[3] of 10 tons capacity is shown in Figure 110. It consists of a heavy cast-iron truck, carrying a steel slag receiver surmounted by cast-iron plates. The cast-iron plates continue down inside the car to near the bottom. This is brick-lined to prevent any iron that may have come out with the slag from burning a hole into it. The top is held in place by a pin (not shown). It is discharged by means of a pole extending outward from the locomotive. The tilting has to be done quickly in order that the skull may be floated out by the cinder.

(3) *Weimer Gondola Cinder Car*[4] is shown in Figure 111. It is a steel tub of 100–260 cubic feet capacity, lined with fire brick or cast iron, and suspended from two trucks. The manner of tilting is seen in the drawing.

The Ramsay Slag-Pot Cleaning Device, Figure 112,[5] has been constructed to remove mechanically any skulls, *H*, that may remain in the Weimer ladle *B*, instead of doing it by means of a pick, bar, sledge, etc. In the bottom of the pot is an opening, through which passes the lug *U* of a plate or disk *C*, forming a false bottom. When a skull forms, this plate *C* is raised by the locomotive pulling on the rope *G*, which is hooked onto the bar *E*, and passing over the pulleys, *W* and *Y*, raises the rod *D* and with it the plate *C* and the skull. At Ensley, Alabama, this device has effected a saving of $5 a day in labor alone, and a ladle has been constantly in use for nine months without requiring any repairs.

Other ladles are those of Treadwell,[6] Tread-Hill,[7] Stewart-Eaton.[8]

(4) *The Howdon Endless-chain Slag Machine*,[9] which is used at the Cambria Iron Company's Works, Johnstown, Pennsylvania, is shown in Figure 113. It is similar to the pig-casting machines previously described. *AA* represents a pair of endless link-belt chains; *B* is the primary driven shaft; *C* and *E* are two pairs of pulleys 100 feet apart, *C* being driven from *B* by gearing, and the block of *E* being connected by links, *N*, *M*, and *L*, with the worm wheel and screw fixed at *f* so as to permit tightening up the chain. There are ninety shallow pans, one overlapping the other, for carrying the slag. They are fixed to the chain by bolts, and are made of three pieces. *F* is a slag trough; *D* a water trough 25 feet long, the water being

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[1] "Iron Age," December 24, 1891.

[2] *Op. cit.*, July 23, 1891.

[3] *Op. cit.*, December 27, 1894.

[4] "Transactions American Institute of Mining Engineers," XXVII, p. 21; "American Manufacturer," 1903, p. 163.

[5] "American Manufacturer," March 24, 1899.

[6] "Iron Age," July 3, 1902.

[7] *Op. cit.*, May 29, 1902.

[8] "Iron Trade Review," June 4, 1903.

[9] "Iron Age," March 10, 1892; "Transactions American Institute of Mining Engineers," XXVII, pp. 23, 24, 39.

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FIG. 118.

GRAPHIC REPRESENTATION OF THE CHEMISTRY OF THE BLAST FURNACE.

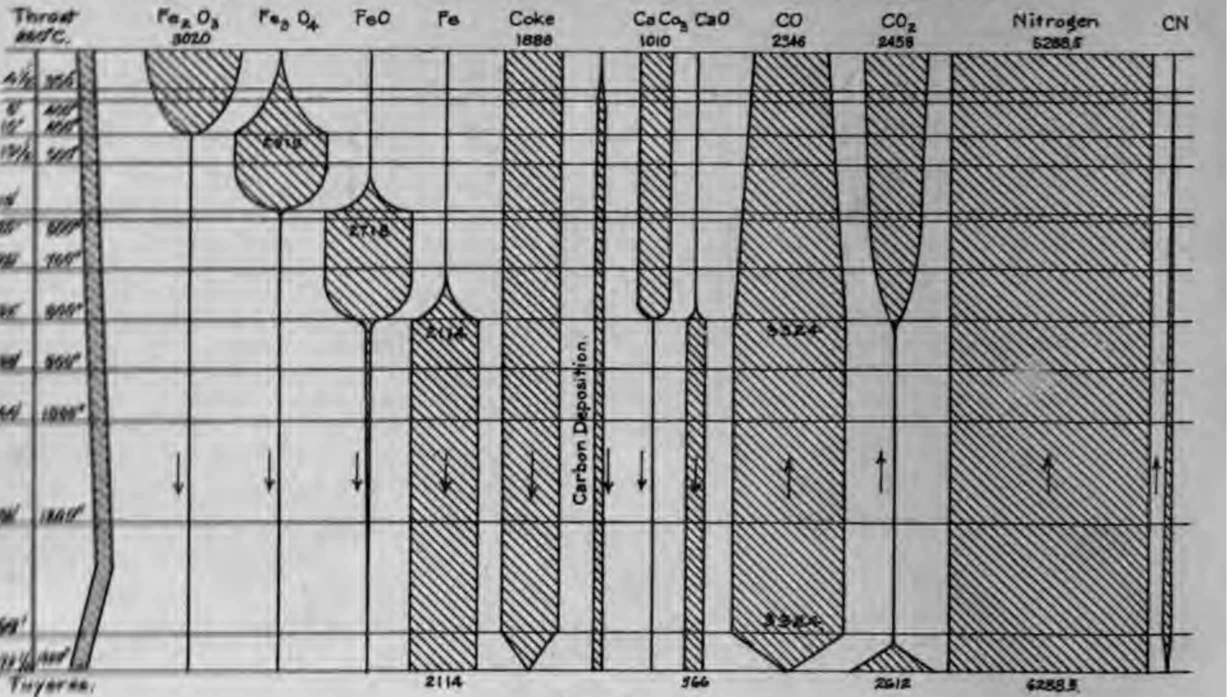
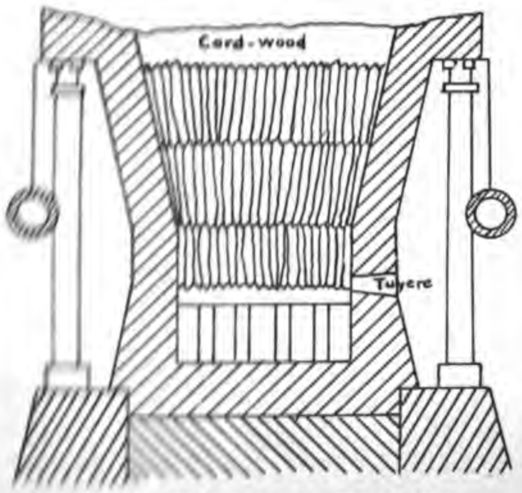


FIG. 118.

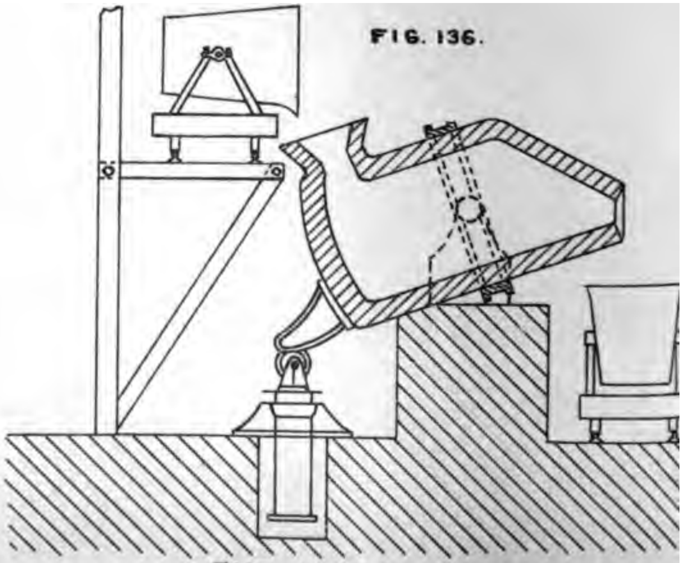
Figures represent lbs. per 2240 lbs. of Pig Iron, Horizontal Distances are proportional to weights.

FIG. 122



BLOWING IN OF DURHAM FURNACE.

FIG. 136.



THE HAERCH MIXER.

kept at a level which allows it to reach only two-thirds up the sides of the pans; *H* is a perforated pipe for sprinkling; *H* a sprinkler; *F'* a railroad car. The speed of the pan is about 13 feet per minute; the slag is run into the pans in a stream 1 to 2 inches in thickness; on chilling it breaks into pieces of from nut-size to those weighing several pounds. It is used as road material. At the Newport works eight machines handle 1,000 tons slag in twenty-four hours.

SECTION 20. **Chemistry of the Blast Furnace.** In a blast furnace there are two currents passing in opposite directions. A rapid gas current, with a high temperature (1,600° C.) and a strong reducing power, rises from the level of the tuyères and passes off at the throat of the furnace cooled down to 200–300° C. It has given off its heat to the slowly descending charge current, and has acted chemically upon it. The Fe_xO_y in the charge in its descent is reduced to Fe. This is carburized and melted; the gangue, with the fluxes, has combined to a melted slag (SiO_2, xCaO, $y Al_2O_3$), and the two are tapped at different levels of the hearth.

A. *Carbon, CO, and* CO_2. Cold or hot blast enters the furnace through the tuyères at a pressure of 4–25 pounds per square inch, and striking incandescent carbon oxidizes it.

The oxidation of C begins at 400° C., and the affinity of C and O increases with the temperature.

(1) $C_2 + 2O_2 = 2CO_2$ (+ 193,920 cal., 1 kilogram C evolving 8,080 cal.) takes place at a low temperature or when C acts upon a readily reducible oxide.

(2) $C_2 + O_2 = 2CO$ (+ 57,240 cal., 1 kilogram C evolving 3,473 cal.) takes place at a high temperature or when C acts upon a difficultly reducible oxide.

Thus Ernst,[1] allowing dry air to act upon coke at different temperatures, found that at 1,000° C. all the C had burned to CO. Thus at the level of the tuyères, where there is a temperature of about 1,600° C., all the C will burn to CO.

Temperature, ° C.	VOLUMES.			$\frac{CO}{CO_2}$
	O	CO_2	CO	
394	20.1	1.1	0.0	0.00
495	0.0	19.0	1.6	0.08
800	0.0	17.9	5.9	0.33
900	0.0	10.1	15.8	1.56
995	0.0	0.0	34.4	. . .

(3) $CO + O = CO_2$ (+ 68,320 cal., 1 kilogram CO evolving 2,400 cal.). The reducing action of CO begins at 200° C., increases to about 1,000° C., and then decreases; hence below 1,000° C. (or in the upper part of the blast furnace) it is a strong, above 1,000° C. (or in the lower part of the blast furnace) it is a weak reducing agent.

(4) $CO_2 + C = 2CO$ (− 41,570 cal., 1 kilogram CO_2 evolving − 945 cal.). The reduction of CO_2 by C begins at 550° C. and is practically complete at 1,000° C. if continued for

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[1] "Journal für praktische Chemie," 1893, LVI, p. 31.

a sufficient length of time. Naumann-Pistor,[1] conducting CO_2 over charcoal, found that there had been formed 12.6, 58.3, 92.2 volumes CO at 550°, 900°, 1,000° C.

The more recent investigations of Boudouard[2] show that while the reduction of CO_2 by C is a function of time and temperature, and while the reduction progresses steadily, it is never wholly complete, but stops on account of an equilibrium being formed between CO_2 and CO. Thus:—

At 650° C. reduction stops when the gas has been changed to 61.5% CO_2 + 38.5% CO.
At 800° C. reduction stops when the gas has been changed to 7.0% CO_2 + 93.0% CO.
At 925° C. reduction stops when the gas has been changed to 4.0% CO_2 + 96.0% CO.

The speed of the reaction is greater at 800° C. than at 650° C., and is influenced by the porosity of the carbon or its fine state of division. Boudouard's results are represented graphically in Figure 114. At 650° and 800° C. deposited carbon (see next paragraph) does not act as energetically as charcoal; at 800° C. dense retort charcoal is less active than coke, and this still less than deposited carbon and charcoal.

(5) *Carbon Deposition.* Carbon monoxide conducted over a heated catalyzer is split into carbon and carbon dioxide, $2CO = C + CO_2$ (+ 39,360 cal.). This reaction is the reverse of the one given in the preceding paragraph. It must, therefore, also be a function of time and temperature. The amount of CO_2 formed increases steadily, but the decomposition of CO is incomplete on account of an equilibrium being formed. Thus Boudouard,[3] using Fe_2O_3, NiO, CaO as catalyzers, found, with Fe_2O_3, that:—

At 445° C. decomposition stopped when the gas had been changed to 98% CO_2 + 2% CO.
At 650° C. decomposition stopped when the gas had been changed to 61% CO_2 + 39% CO.
At 800° C. decomposition stopped when the gas had been changed to 7% CO_2 + 93% CO.
At 900° C. decomposition stopped when the gas had been changed to 3.5% CO_2 + 96.5% C.

The speed of the reaction is greater at 800° C. than at 650° C., and increases with the amount of catalyzer used. Graphically these results are shown in Figure 115.

The equations given under (4) and (5) are therefore reversible: $CO_2 + C \rightleftarrows CO$. The conditions given under (4) show the reaction to proceed from left to right; those given under (5) make it act from right to left. The reaction either way stops when the resulting gas with a given temperature reaches a certain composition.

Schenck and Zimmermann[4] maintain that an oxide cannot act as a catalyzer, a metal being required, and that the reaction $2CO = C + CO_2$ is made up of two separate steps, $CO = C + O$ and $CO + O = CO_2$, the nascent O showing a strongly oxidizing power.

By iron metallurgists it has always been maintained and is still held that carbon deposition begins when CO is conducted over Fe_2O_3 heated to 250° C.: $2Fe_2O_3 + 8CO = 7CO_2 + 4Fe + C$ (the equation shows that some iron oxide is reduced), and stops at 600° C. The deposited C is taken up by the Fe at a temperature approaching that at which the slag begins to form. The iron, in addition to the deposited C, absorbs C from the coke. By this absorption the melting-point of the iron is lowered to that of cast iron, viz., 1,100–1,250° C. According to Bell[5] the

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[1] "Berichte der deutschen Chemischen Gesellschaft," 1885, XVIII, p. 1647.

[2] "Annales de Chimie et de Physique," 1901, XXIV, p. 28.

[3] *Op. cit.*, p. 50.

[4] "Berichte der deutschen Chemischen Gesellschaft," 1903, XXXVI, p. 1231.

[5] "Principles," p. 192.

carbon deposition ceases when there are two volumes CO_2 to one volume CO. This proportion must, however, vary with the temperature, as the oxidizing effect of CO_2 increases with the temperature. Lump ore[1] thus exposed at an elevated temperature to a current of CO (or blast-furnace gas) swells, cracks, and becomes disintegrated and greatly increases in volume. Soft hematites showed 13.82 per cent. deposited C on the original weight of the ore, hard hematite 7.52 per cent., blue ores 3.08 per cent., brown ores 12.94 per cent., magnetite practically no C, cinders and scales 0.38 per cent., and Mesabi ores 21.61 per cent. The large amount of deposited C with Mesabi ores, increasing them to four to five times their original volume, explains some of the troubles encountered in their use.

B. *Reduction of Fe_xO_y by CO, C, Cy, and N.*[2] The reduction of Fe_2O_3 by CO to a lower state of oxidation begins at 200° C.,[3] but the action is weak; at 400-450° C. it is readily reduced to Fe_3O_4,[4] and, if given enough time, to Fe_6O_7; the temperature must be higher if FeO is to result; according to Tunner[5] it must reach 900° C. with average ores if metallic iron is to result; Wiborgh (*l. c.*) obtained it with readily reducible ores at 700° C.[6] In order to reduce the iron oxide to metallic iron a temperature of at least 800° C. must be maintained.

The reduction of Fe_2O_3 by C begins at about 400° C. and grows stronger with increase of temperature.

Reduction of Fe_2O_3 by CO gives exothermic reactions:—

$$\begin{aligned} Fe_2O_3 + 3CO &= 2Fe + 3CO_2 \ (+\ 4.6 \text{ cal.}) \text{ begins at } 200^\circ \text{ C.} \\ Fe_3O_4 + 4CO &= 3Fe + 4CO_2 \ (+\ 6.2 \text{ cal.}) \text{ begins above } 450^\circ \text{ C.} \\ FeO + CO &= Fe + CO_2 \ (+\ 1.6 \text{ cal.}) \text{ begins at } 700^\circ \text{ C. } (?). \end{aligned}$$

Reduction of Fe_2O_3 by C gives endothermic reactions:—

$$\begin{aligned} Fe_2O_3 + 3C &= 2Fe + 3CO \ (-112.4 \text{ cal.}) \text{ begins at } 400^\circ \text{ C.} \\ Fe_3O_4 + 4C &= 3Fe + 4CO \ (-149.8 \text{ cal.}) \text{ begins at } 645^\circ \text{ C.} \\ FeO + C &= Fe + CO \ (-\ 37.4 \text{ cal.}) \text{ begins at } 685^\circ \text{ C.} \end{aligned}$$

Hence the aim must be to reduce as much Fe_xO_y as possible by means of CO. But the resulting CO_2 acts as an oxidizing agent:—

$$\begin{aligned} 2FeO + CO_2 &= Fe_2O_3 + CO \ (+\ 192.6 \text{ cal.}) \\ 2Fe + 3CO_2 &= Fe_2O_3 + 3CO \ (-\ 4.6 \text{ cal.}) \end{aligned}$$

unless diluted by CO, and the higher the temperature the more it must be diluted. The reduction of Fe_xO_y to metallic Fe is not completely effected by CO alone, as[7] there is a tendency for the iron to be reoxidized and for C to be separated:—

$$\begin{aligned} Fe + CO &= FeO + C \ (+\ 37.4 \text{ cal.}) \\ 4Fe + 3CO_2 &= 2Fe_2O_3 + C \ (+\ 107.8 \text{ cal.}) \end{aligned}$$

the deposited C finishing the reduction.

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[1] Laudig-Bachman, "Transactions American Institute of Mining Engineers," XXVI, pp. 269, 1061.

[2] Tholander, "Oesterreichisches Jahrbuch," 1878, p. 55; Kosmann, "Stahl und Eisen," 1888, p. 586.

[3] Bell, "Principles," p. 191.

[4] Wiborgh, "Stahl und Eisen," 1897, pp. 804, 858.

[5] "Oesterreichisches Jahrbuch," IX, p. 309.

[6] See also Braithwaite, "Chemical News," 1895, LXXII, p. 211.

[7] Bell, "Principles," p. 188.

Baur and Glaessner[1] have drawn curves which give the conditions under which the reversible equations $FeO + CO \rightleftarrows Fe + CO_2$ and $Fe_3O_4 + CO \rightleftarrows 3FeO + CO_2$ are in a condition of equilibrium. In Figure 116 the abscissa represents temperatures 330–990° C., the ordinate percentages of CO_2 (the rest being CO), and the full-drawn lines the equilibrium curves for Fe, FeO, and FeO, Fe_3O_4, showing maxima at 680° and 490° C. A gas of the composition $60CO_2 + 40CO$ at 550° C. will have an oxidizing effect upon FeO or Fe_3O_4, while one of the composition $50CO_2 + 50CO$ will act as a reducing agent. The two dotted lines represent Boudouard's carbon-deposition curves[2] at pressures of 1 and of $\frac{1}{4}$ atmosphere. Each cuts the curves in two points. Considering the curve to the right: its cutting the equilibrium curves at two points means that at only two points (685° C. and 645° C.) can deposited carbon be in equilibrium with Fe, FeO, and FeO, Fe_3O_4 in a gas mixture of CO and CO_2. If in the upper curve carbon be added at a temperature *below 685° C.*, the equilibrium will be disturbed and a reaction set in to oxidize one of the solids Fe or C. It will be the Fe, because in equation $2CO \rightleftarrows C + CO_2$ equilibrium can obtain only after that of $FeO + CO \rightleftarrows Fe + CO_2$, with its larger amount of CO_2, has been disturbed. The disturbance will continue until all the Fe has been oxidized. A similar oxidation will occur in the FeO, Fe_3O_4 curve if C be added at a temperature *below 645° C.*, the FeO being oxidized. If in the upper curve C be added at a temperature *above 685° C.* the conditions will be reversed, as equation $FeO + CO \rightleftarrows Fe + CO_2$ demands more CO_2 to be kept in equilibrium than $2CO \rightleftarrows C + CO_2$ can bear, hence the reaction will proceed in the sense $C + CO_2 \longrightarrow CO$. If in the lower curve C be added at a temperature *above 645° C.* the CO will have a reducing effect, first upon Fe_3O_4, then upon FeO, and finally metallic iron will be formed. From what has been said it can be seen that furnace troubles from deposited C[3] can occur only below 685° C., and will be cured if the temperature is raised above 685° C.[4]

The reduction of silicates of iron is effected only by C and at a high temperature.

Cyanides are formed in the blast furnace[5] by the action of N upon incandescent C and combine with K or Na of the charge. The resulting alkaline cyanides are volatilized and ascend with the gas current. In the upper part of the furnace they act upon iron oxide.[6]

$2KCN + 3FeO = K_2O + 2CO + 3Fe + 2N$, whereby C and N are restored to the gases. Some of the alkali is carried off with the gases, some combines with CO_2, descends with the charges, and passes off with the slag, or, combining again with CN, repeats the cycle of motion. The action of CN in the blast furnace is of minor importance.

Hydrogen is set free by the moisture of the air acting upon incandescent carbon. $H_2O + C = H_2 + CO$ (− 29,535 cal.). It plays an unimportant rôle in the reduction of iron oxide [$Fe_2O_3 + H_6 = Fe + 3H_2O$ (− 26.5 cal.)]. Its main importance lies in the reduction of temperature at the tuyère level on account of the absorption of heat due to the decomposition of H_2O. In fact, when the air is moist more coke will be required than when it is dry. For

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[1] "Zeitschrift für Physikalische Chemie," 1903, XLIII, p. 354; "Stahl und Eisen," 1903, p. 556.

[2] See also "Bulletin de la Société Chimique," XXIII, p. 137.

[3] Roberts, "Iron Age," May 19, 1904.

[4] See "Irregularities in the Furnace," p. .

[5] "Berg- und Hüttenmännische Zeitung," 1904, p. 184.

[6] Bell, "Principles," pp. 217, 223.

this reason the Maryland Steel Company, located on the shore near Baltimore, Maryland, measures daily the moisture of the air. At the Isabella furnace, Pittsburgh, Pennsylvania, experiments are under way[1] to deprive the air of its moisture before it enters the blowing engine by passing it through a refrigerating machine.

C. *Reduction of Mn, Si, P, S, Cu (Ti, Cr, Va).* All cast iron contains more or less Mn. MnO_x can be reduced to Mn only by solid C and at a high temperature, $MnO_x + xC = Mn + xCO$; the reduction is facilitated by the presence of Fe (with which it readily alloys) and by a basic slag (as MnO readily combines with SiO_2). It is not possible to reduce all the Mn in the blast furnace. Some always enters the slag and the proportion increases with the percentage of Mn in the pig. Manganese[2] melts at 1,245° C.; it becomes incandescent and burns when brought in an atmosphere of N to 1,210–1,220° C.

Silica also can be reduced to Si only by solid C and at a high temperature, about 1,200° C.[3] $SiO_2 + 2C = Si + 2CO$ (– 162.0 cal.), and requires the presence of Fe to absorb it. It is more difficult of reduction than MnO_x. Usually not more than one-twentieth of Si in the charge is reduced; the rest goes into the slag.

Phosphorus, which is present in the charge as iron and lime phosphate, requires a high temperature and solid C to be reduced to a phosphide, $P_2O_5 + C_5 = P_2 + 5CO$ (– 85.9 cal.). The reduction is favored by an acid slag, but even with a basic slag all of the P enters the pig on account of its strong affinity for Fe.

Sulphur occurs in the charge as sulphide, sulphate, and organic (from coke) sulphur. Usually not more than one-twentieth of it enters the pig, the rest being taken up by the slag as CaS, for which purpose special additions of limestone are made.[4]

Copper is readily reduced and all goes into the pig.

Titanium, chromium, and vanadium, difficult of reduction, go only to a small extent into the pig iron.

D. *Iron Blast-furnace Slag.* See Section 22.

E. *Changes in the Ascending Current of Gases.* The blast, entering through the tuyères, strikes C heated to about 1,600° C. and oxidizes it wholly to CO. Any small amounts of CO_2 formed are instantly converted by the C to CO. The water vapor is converted into $H_2 + CO$.

(1) The ascending current then consists of approximately 64 volumes N, 34CO, and 2H.[5]

(2) Rising in the furnace, the percentage of CO is increased by the reduction of Si, Mn, and Fe through solid C; any CO_2 formed by the action of CO upon Fe is quickly converted back again to CO by C. The increase of CO continues until the temperature is reached at which CO begins to act as a reducing agent, say at 1,000° C.

(3) Now the percentage of CO_2 is increased, at first slowly and then more rapidly; slowly because the weak action of CO forming CO_2 is opposed by the strong tendency of C to reduce it again to CO; but with decrease of temperature the power of CO rises and that of C sinks

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[1] Gayley, United States Patent, No. 756,110, March 29, 1904.

[2] "Zeitschrift für Electrochemie," 1902, VIII, p. 186.

[3] *Op. cit.*, 1902, VIII, p. 186.

[4] Elbers, "Engineering and Mining Journal," January 19, 26, [illegible] 1903, p. 163.

[5] Van Vloten, "Stahl und Eisen," 1893, p. 28.

and finally stops. The percentage of CO_2 is further increased by the liberation of CO_2 from limestone ($CaCO_3 = CaO + CO_2$) which occurs at about 800° C.

(4) Further up the $2CO$ is decomposed into $CO_2 + C$, again increasing the percentage of CO_2.

(5) Finally, the gases take up H_2O from the ore and pass off at a temperature of 200–300° C. through the downcomer. Their composition has been changed, after deducting the water vapor, to say N 60 volumes, CO 24, CO_2 12, H 2, CH_4 2. The marsh gas comes from the residual volatile matter of the coke. Bell[1] analyzed the gases from different levels of a coke blast furnace, 80 feet high and 25,500 cubic feet capacity, smelting Cleveland (England) ironstone (carbonate ore).

	HEIGHT ABOVE TUYÈRES.					
	2 feet.	26½ feet.	39¾ feet.	45½ feet.	58¼ feet.	76 feet.
CO_2 . . .	1.90	0.46	0.00	3.84	6.56	10.69
CO . . .	39.18	35.80	34.82	32.33	32.25	28.58
H	1.97	1.14	1.41	3.88	1.08	1.70
N	56.95	62.60	63.77	59.95	60.11	59.03
$\frac{CO_2}{CO}$	0.05	0.01	∞	0.12	0.20	0.38

Rinman obtained in a charcoal furnace[2] 40.8 feet high, smelting roasted magnetite, the following data:—

	HEIGHT ABOVE TUYÈRES.			
	7¼ feet.	14¾ feet.	23½ feet.	34 feet.
Volumes O	0.8	0.0	8.0	14.2
Volumes C	24.4	25.5	28.9	31.2
$\frac{CO_2}{CO}$	0.18	0.03	0.24	0.44

The [illegible] $\frac{CO_2}{CO}$ is of importance. T[illegible] the percentage of CO_2 (leaving out CO_2 from [illegible] ter has been the [illegible] reduction by CO and the smaller the direct [illegible] e furnace, *i.e.*, the smaller the total fuel

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[illegible] nde," II, p. 226.

[illegible]

[illegible]

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

consumption. If all the iron were reduced by CO, the ratio would be 1.217.[1] The limit in practical work is 0.3–0.8; commonly it runs from 0.4 to 0.5. Whiting[2] records, in his trial runs at the Illinois Steel Company, 0.66 with a relatively low fuel consumption.

(6) It is known that no saving of fuel is effected by increasing the height of the blast furnace beyond a certain limit. This limit was with coke furnaces formerly 80 feet. It has been raised in this country to 106 feet, but has not been found economical on account of fuel consumption. Furnaces built at present are about 80 feet high. The reason for limiting the height is that as long as the furnace is high enough for the reduction of Fe_2O_3 to Fe by CO, an addition to the height will expose the descending coke for a longer time than necessary to the oxidizing effect of CO_2 (beginning at 550° C.) in the ascending gas current, with the result of an increased reduction to CO and a consequent unnecessary gasification of coke. The curve in Figure 117[3] shows how, with increase of capacity and corresponding height of a blast furnace, the temperature of the waste gases decreases rapidly at first, then more slowly, and finally not at all. The denser the blast-furnace fuel the less it is attacked by CO_2[4] and the higher can be the furnace. Furnaces using anthracite are, however, lower than coke furnaces, on account of the anthracite breaking up and offering too great a resistance to the passage of the gases and causing other troubles; charcoal furnaces are lower than coke furnaces.

(7) The weight of gas passing off from the furnace can be found either by measuring the velocity and temperature in the downcomer of known area and determining the specific gravity from the chemical analysis, or by calculation[5] from the amount of blast introduced and the amount of C gasified per minute, the chemical analysis showing how the C is distributed in the gas. Thus a blast furnace producing in twenty-four hours 200 tons pig (4 per cent. C), with a coke consumption of 1 ton of coke (80 per cent. C) per ton of pig, will gasify in twenty-four hours (C pounds in coke + C pounds in limestone + C pounds in ore) − C pounds in pig = A pounds C, or, per minute, $\frac{a}{24 \times 60}$ pounds C. From the chemical analysis of the gas the distribution of the C can be calculated, and from this, if desired, the volume composition.

F. *Changes in the Descending Ore Charge.* As regards the changes in the descending ore charge, four zones can be distinguished: the zone of preparatory heating; the upper zone of reduction; the lower zone of reduction; the zone of fusion. There is no sharp line of demarcation between these zones, one passing over into the other, the reactions beginning in one zone being only finished in the next, or perhaps even the second one lower down. The process of heat interception is a continuous one, extending from the tuyères to the throat. The decrease in temperature in the blast furnace is shown by the following measurements of Wiebmer[6] at a coke blast furnace at Gleiwitz, Silesia, the dimensions of which were: height from tuyères to throat, 44.6 feet; hearth, 8.4 feet; bosh, 17.5 feet; throat, 12.8 feet; capacity, 7,592 cubic feet; the blast delivered, 5,297 cubic feet air of 350° C. per minute; the ore was limonite, siderite, and puddle cinder (table, p. 72).

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[1] Grüner-Gordon, "Studies in Blast-furnace Phenomena," Philadelphia, Baird, 1874, p. 33.

[2] "Transactions American Institute of Mining Engineers," XX, p. 290.

[3] Bell, "Principles," p. 75.

[4] Bell, "Journal Iron and Steel Institute," 1872, I, p. 74; Åkerman, "Oesterreichisches Jahrbuch," 1889, p. 127.

[5] Osann, "Stahl und Eisen," 1901, p. 906.

[6] "Zeitschrift für Berg-, Hütten-, und Salinen-Wesen in Preussen," 1874, XXII, p. 289.

	HEIGHT ABOVE TUYÈRE LEVEL.								
	0 feet.	1.4 feet.	4.8 feet.	18.2 feet.	25.0 feet.	32.2 feet.	39.4 feet.	44.6 feet.	
At centre	1,300	1,400	1,400	1,200	955	850	680	140	° C.
Halfway between .	1,500	1,500	1,300	1,000	700	525	432	to	° C.
At wall	1,600	1,300	1,400	1,200	900	815	575	290	° C.

(1) The Zone of Preparatory Heating (100–300° C.). The charge when fed into the furnace gives up its hygroscopic water, then that which is chemically combined; this absorbs considerable heat, and keeps the top of the furnace comparatively cool. Limonites often give up the last of their chemically combined water only at a temperature of 500° C.; thus reduction may have begun on the surface before all the water has been removed from the centre.[1]

(2) The Upper Zone of Reduction (300–900° C.). Here CO is the reducing agent. Fe_2O_3 slowly gives up its O until at about 800° C. it has been completely reduced to spongy Fe. At 250° C. begins the carbon deposition, and continues until about 600° C. Thus the two reactions go on simultaneously, starting at a surface of the ore and slowly penetrating to the centre. At 800° C., however, $CaCO_3$ gives up its CO_2, which is more or less reduced to CO by C, and prevented from attacking the reduced Fe by the deposited C. Siderites which[2] begin to give up their CO_2 at 300–400° C. yield it completely only at 800° C.[3] As the reducing reaction of CO is exothermic, an upper zone of reduction coexists with an upper zone of heat generation.

(3) The Lower Zone of Reduction (900–1,200° C.). Here C is the reducing agent. While the action of C begins at 400° C., its effect becomes strong only when that of CO becomes weak. C will finish the reducing work of CO upon Fe_xO_y, and do its specific work in reducing MnO_x, SiO_2, P_2O_5, and slagged iron and manganese. These processes continue some distance into the next zone. With the reduction of iron by C begins the absorption of C, which continues until the melted cast iron has passed the tuyère level.

(4) Zone of Fusion (1,200–1,600° C.). This should begin only when all Fe_xO_y has been reduced, otherwise FeO will be slagged. At the beginning of the zone of fusion, parts of the unreduced charge sinter together, others become thoroughly melted, and lower down the latter dissolve, more or less, the former, until a short distance above the tuyères a slag of approximately uniform composition has been formed. This trickles down through the coke which fills the zone of fusion. The cast iron in its downward course through the coke takes up more and more C. Cast iron and slag, having attained the desired composition, collect in the hearth and separate according to their specific gravities.

Graphically the changes in the ascending gas current and the descending ore charge are shown in Figure 118,[4] the horizontal width representing roughly the amounts of the

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[1] Ebelmen, Percy-Wedding, "Eisenhüttenkunde," II, p. 659.

[2] Åkerman-Turley, "Das Rösten der Eisenerze," p. 2.

[3] Schinz, "Documents," p. 70.

[4] This diagram was conceived by R. H. Richards ("Notes on Iron," p. 39), was then used with additions by the writer (1st ed. of these notes, p. 77), and lastly published by H. H. Campbell ("The Manufacture of Iron and Steel," 2d ed., 1903, p. 71), with Dougherty's changes, which made quantitative what had been only qualitative.

products. The conditions assumed for diagram are: height of furnace, 90 feet; temperature at tuyères, 1,500° C.; no water; coke (87 per cent. C), 1,800 pounds per ton of iron; limestone (100 per cent. $CaCO_3$), 1,010 pounds per ton of iron; pig iron, 4 per cent. C and 1 per cent. Si; waste gas, $1CO_2 : 1\frac{1}{2}CO$, and 260° C.

SECTION 21. **Estimate of the Quantity of Heat Required in Iron Smelting.** The quantity of heat required in iron smelting varies to some extent according to circumstances. It consists of the heat really absorbed by each stage of the operation, comprising chemical action and fusion, and the heat lost. The following tables, taken from H. H. Campbell's volume just quoted, are elaborations of the calculations of Bell.[1]

COMPARISON OF FURNACE PRACTICE AT MIDDLESBROUGH, ENGLAND, AND PITTSBURGH, PENNSYLVANIA.

	Middlesbrough.	Pittsburgh.
General conditions: —		
Height of furnace, feet	80	80
Cubic contents, feet	25,500	18,200
Per cent. of metallic iron in ore	39.0	59.0
Weekly product per 1,000 feet cubic contents, tons	21.57	128.00
Temperature of blast, ° C.	704	593
Temperature of tunnel-head gases, ° C.	250	171
Ratio of CO to CO_2 in gases	2.11	2.35
Data per ton of pig iron: —		
Coke, pounds	2,239	1,882
Limestone, pounds	1,232	1,011
Ore, pounds	5,376	3,613
Weight of blast, pounds	9,761	7,974
Weight of tunnel-head gases, pounds	13,381	11,211
Slag, pounds	3,136	1,200
Calories used in the furnace per ton of pig iron: —		
Reduction of Fe_2O_3	1,681,887	1,681,887
Reduction of metalloids in pig iron	212,039	133,655
Dissociation of CO	73,152	74,168
Fusion of pig iron	335,280	335,280
Evaporation of water in coke	13,970	4,216
Decomposition of water in blast	120,904	118,516
Expulsion of CO_2 from limestone	206,756	157,175
Reduction of this CO_2 to CO	214,579	177,190
Fusion of slag	782,320	299,212
Radiation, cooling water, etc.	494,792	298,145
Total absorbed in furnace	4,135,679	3,279,444
Calories in tunnel-head gases per ton pig iron: —		
Sensible heat	364,000	254,700
Potential as CO	3,810,000	3,137,000
Total in tunnel-head gas	4,174,000	3,391,700
Summary per ton of pig iron: —		
(*a*) Calories used in furnace (as above)	4,135,679	3,279,444
(*b*) Calories in tunnel-head gases (as above)	4,174,000	3,391,700
Sum of (*a*) and (*b*)	8,309,679	6,671,144
(*c*) Less calories from blast included in (*a*)	738,632	626,872
Calorific power produced per ton of iron	7,571,047	6,044,272
Calorific power produced per ton of coke	7,574,400	7,196,000

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[1] "Transactions American Institute of Mining Engineers," XIX, p. 957.

DISTRIBUTION OF CALORIFIC ENERGY ON THE ASSUMPTION OF THE SAME COKE FOR MIDDLESBROUGH AND PITTSBURGH.

	EQUIVALENT IN POUNDS OF COKE.[1]		PER CENT. OF TOTAL CALORIFIC VALUE.	
	English.	American.	English.	American.
Constant factors: —				
Reduction of Fe_2O_3	452	452	20.2	25.2
Fusion of pig iron	90	90	4.0	5.0
Total	542	542	24.2	30.2
Factors beyond the control of the smelter: —				
Reduction of the metalloids	58	36	2.6	2.0
Expulsion of CO_2 from limestone	56	41	2.5	2.3
Reduction of this CO_2 to CO	58	49	2.6	2.7
Fusion of slag	210	80	9.4	4.5
Total	382	206	17.1	11.5
Factors more or less under control: —				
Dissociation of CO	20	20	0.9	1.1
Evaporation of water in coke	5	2	0.2	0.1
Decomposition of water in blast	34	33	1.5	1.8
Radiation, cooling water, etc.	134	80	6.0	4.5
Total	193	135	8.6	7.5
Tunnel-head gases: —				
Sensible heat	99	68	4.4	3.8
Potential as CO	1,023	837	45.7	47.0
Total	1,122	905	50.1	50.8
Grand total	2,239	1,788	100.1	100.0

[1] The preceding table shows that the English coke was 5 per cent. better than the American coke. Hence with the same coke the fuel in Pittsburgh would have been 1,788 pounds per ton instead of 1,882.

GENERAL EQUATION OF THE BLAST FURNACE.

	Middlesbrough.	Pittsburgh.
Per ton of pig iron: —		
Calories from formation of CO_2	2,427,000	1,982,000
Calories from formation of CO	1,336,000	1,025,000
Calories potential in gas as CO	3,810,000	3,137,000
Total per ton of iron	7,573,000	6,144,000
Per ton of coke: —		
Calories from formation of CO_2	2,428,000	2,360,000
Calories from formation of CO	1,342,000	1,220,000
Calories potential in gas as CO	3,812,000	3,735,000
Total per ton of coke	7,582,000	7,315,000
Distribution by per cent. of total energy: —		
Per cent. from formation of CO_2	32.1	32.2
Per cent. from formation of CO	17.6	16.7
Per cent. potential in gas as CO	50.3	51.1
Total	100.0	100.0

Wedding[1] gives the following as an average for coke furnaces: —

(1) Evolution of Heat: —		
by formation of CO_2	40 per cent.	
by formation of CO	40 per cent.	
by heat in blast	20 per cent.	
		100 per cent.
(2) Heat Required for Chemical Action and Fusion: —		
for reduction	40 per cent.	
for evaporation and decomposition	11 per cent.	
for fusion of pig	6 per cent.	
for fusion of slag	9 per cent.	
(3) Losses: —		
by radiation, convection, etc.	27 per cent.	
by escaping gases	7 per cent.	
		100 per cent.

For detail calculations see Bell,[2] Ledebur-Nitze,[3] Billy,[4] Billy-Stoek,[5] Bjerregard,[6] Jüptner,[7] Wedding,[8] Osann,[9] Waldeck.[10]

In "Burdening" the blast furnace, *i.e.*, giving the proper amount of fuel to charge, the furnace man makes the ratio of pig produced to fuel consumed his standard. In the coke blast furnace the ratio of 1 ton of coke : 1 ton of pig is not uncommon. Gayley[11] gives as the lowest coke consumption for 1889 at the Edgar Thomson Works 1,920 pounds per long ton of pig. According to Potter[12] a furnace of the Union Plant of the Illinois Steel Company has made over 5,800 tons of Bessemer pig iron a month with a fuel consumption of 1,600 pounds Connellsville coke per 1 long ton of pig. An average of four furnaces of the Illinois Steel Company in 1887 for one month was 1,834 pounds coke per 1 long ton of pig.[13] Other data are given in the blast-furnace table (facing page 45). These low figures for producing gray iron are due in part to progressive work; in part, however, to the very rich ores that are at the disposal of our smelting works. In Europe, where lower grade ores are treated, the coke consumption is very much greater, reaching 2,300–2,600 pounds per long ton of gray pig. For charcoal and anthracite furnace see blast-furnace table facing page 45.

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[1] "Grundriss der Eisenhüttenkunde," Berlin, 1890, p. 168.

[2] "Chemical Phenomena of Iron Smelting," London, 1872, p. 148; and from his paper, On the Waste of Heat, Past, Present, and Future, in Smelting Ores of Iron: "Journal Iron and Steel Institute," 1893, II, p. 219.

[3] "Engineering and Mining Journal," January 30, 1892.

[4] "Colliery Guardian," December 14, 1894.

[5] "Engineering and Mining Journal," April 11, 1896.

[6] "Iron Age," February 15, 22, 1894.

[7] "Compendium der Eisenhüttenkunde," p. 225.

[8] "Stahl und Eisen," 1892, p. 1029.

[9] *Op. cit.*, 1901, p. 905.

[10] *Op. cit.*, 1903, p. 670.

[11] "Transactions American Institute of Mining Engineers," XIX, p. 956.

[12] *Op. cit.*, XIX, p. 972.

[13] Potter, *op. cit.*, XXIII, p. 373.

FIG. 120

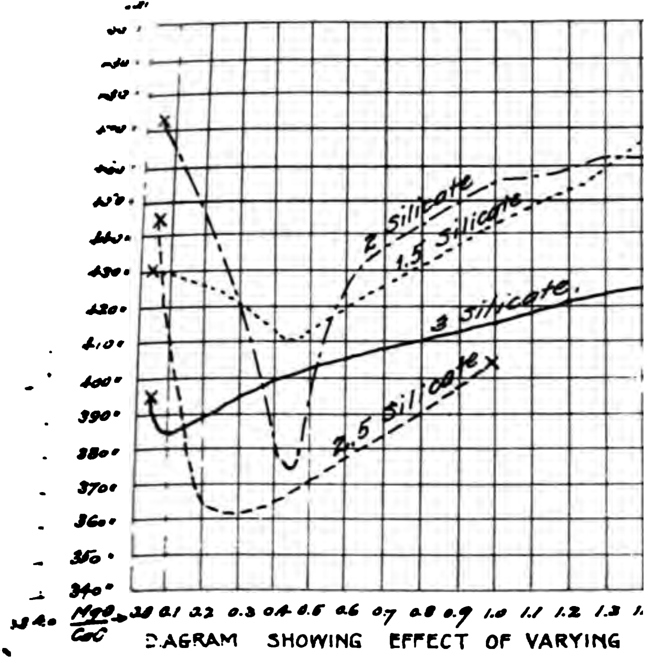

DIAGRAM SHOWING EFFECT OF VARYING
OF CaO MgO ON CERTAIN SILIC

FIG. 121

SECTION 22. **Slags Produced in the Blast Furnace.** In smelting an iron ore the SiO_2, Al_2O_3, and CaO(MgO) of the gangue have to be combined in such a way as to form a fusible slag. Ores that contain these constituents in correct proportions are called self-fluxing; usually one constituent (CaO), occasionally two, are missing, and have to be added (fluxes).

Silicate slags formed in metallurgical operations are classed according to the ratio of O in base to O in acid, as: —

$\frac{\text{O basic}}{\text{O acid}}$	Name.	FORMULA.	
		Univalent base.	Trivalent base.
2 : 1	Subsilicate	$4RO.SiO_2$	$4R_2O_3.3SiO_2$
1 : 1	Singulosilicate	$2RO.SiO_2$	$2R_2O_3.3SiO_2$
2 : 3	Sesquisilicate	$4RO.3SiO_2$	$4R_2O_3.9SiO_2$
1 : 2	Bisilicate	$RO.SiO_2$	$R_2O_3.3SiO_2$
1 : 3	Trisilicate	$2RO.3SiO_2$	$4R_2O_3.9SiO_2$

The univalent bases are CaO and MgO(MnO); the trivalent base is Al_2O_3.

Åkerman[1] studied the melting-points of various SiO_2, Al_2O_3, CaO(MgO) silicates. He melted the silicate in a brasqued crucible, allowed it to solidify partly, and poured the remainder (now actually at the freezing-point) into a calorimeter, and measured in calories the total heat of solidification, *i.e.*, the sum of the latent heat of solidification plus the heat evolved in cooling from the melting-point down.

Figure 119 shows the influence an increase of SiO_2 has on silicates of lime and magnesia free from Al_2O_3. The curves show two minima, one near the 1.5, or sesquisilicate, with about 47 per cent. SiO_2, the other between the 2 and 3, or bi- and tri-silicate, with about 60 per cent. SiO_2.

Figure 120 shows the effects that varying amounts of CaO and MgO have in certain silicates. The lowest melting-point is reached

in a 1.5 silicate when MgO : CaO = 0.44,
in a 2.0 silicate when MgO : CaO = 0.44,
in a 2.5 silicate when MgO : CaO = 0.28,
in a 3.0 silicate when MgO : CaO = 0.1.

Figure 121 shows the effects that additions of Al_2O_3 have on certain calcic silicates. They prove that up to a certain point an addition of Al_2O_3 always lowers the melting-point, after which it raises it again. It is at a minimum

in the 0.5 silicate when Al_2O_3 : CaO = 7 : 5,
in the 0.8 silicate when Al_2O_3 : CaO = 7 : 8,
in the 1.0 silicate when Al_2O_3 : CaO = 1 : 2,
in the 1.5 silicate when Al_2O_3 : CaO = 1 : 45,
in the 2.0 silicate when Al_2O_3 : CaO = 1 : 6,
in the 2.5 silicate when Al_2O_3 : CaO = 1 : 13,
in the 3.0 silicate when Al_2O_3 : CaO = 1 : 30.

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[1] "Stahl und Eisen," 1896, p. 281; Howe, "Transactions American Institute of Mining Engineers," XXVIII, p. 346.

In other words, the lower in SiO_2 a slag the higher must be the Al_2O_3 (as a rule not over 15 per cent.) in order to make the slag readily fusible, *i. e.*, Al_2O_3 acts as an acid. Slags made in coke or anthracite furnaces producing foundry and Bessemer irons are singulosilicates with[1] SiO_2, 30–35 per cent., Al_2O_3, 15–10 per cent., bases 50–55 per cent. Those in charcoal furnaces either lie between the singulo- and bi-silicate, or they are bisilicates and contain SiO_2 45–65 per cent., Al_2O_3 10–5 per cent., bases 45–30 per cent. The reason for the difference lies in the fact that, using expensive ore (rich in Fe and low in S) and expensive fuel (charcoal), smelting must be carried on with as little flux (limestone) as feasible. As the ore is siliceous, the slags made will be siliceous also.

The color of blast-furnace slags when free from Fe and Mn is white; small amounts of Fe and Mn make them gray; large amounts of FeO, say 10 per cent., make them black; of MnO, with little S, greenish. Slags rich in SiO_2 and Al_2O_3 containing little FeO and MnO are liable to be blue. When cooled suddenly the slags are glassy; when slowly, crystalline to stony. Slags are often spongy from the S they contain in the form of CaS[2]:—

$$CaS + 2H_2O + CO_2 = CaCO_3 + H_2S\text{; or}$$
$$2CaS + 4O + CO_2 = CaCO_3 + Ca_2S_2O_3 \text{ and}$$
$$CaS_2O_3 + O = CaSO_4 + S.$$

The specific gravity of slags ranges from 2.5 to 3.0. Acid slags are viscid; basic slags short.

TiO_2, if figured in as a base, makes slags very viscid; if figured as an acid replacing SiO_2, and Al_2O_3 being figured as a base, it makes readily fusible slags.[3]

SECTION 23. **Calculation of Blast-furnace Charge.** In calculating a blast-furnace charge the SiO_2, Al_2O_3, and CaO(MgO) of the ore have to be combined so as to form a fusible slag. The limits of the percentage values have already been given. Further, the ratio of weight of slag to weight of pig has to be considered. The lowest is probably slag : pig = 0.37 : 1, the highest = 2 : 1, and this is the common figure in smelting the rich Lake Superior ores. Ore charges never run under 25 per cent. Fe; in Europe they run 30–40 per cent. Fe, and in this country, when Lake Superior ores are smelted, usually over 40 per cent. Fe.

It has been customary to combine the acid SiO_2 with the bases Al_2O_3 and CaO, according to definite oxygen ratios, making singulo-, sesqui-, and bi-silicate slags. As, however, slags show a great difference in the degree of fusibility according to the relative proportions of CaO and Al_2O_3 to SiO_2, the fusibility decreasing, *e.g.*, rapidly, if with a high percentage of SiO_2 there is present a considerable amount of Al_2O_3, the method has been abandoned in many instances, and the ratio, bases : $(SiO_2 + Al_2O_3)$ made the basis of the calculation.[4] This ratio is usually 51 : 49, or 52 : 48. Within the total of $(SiO_2 + Al_2O_3)$ the proportions are, however, not arbitrary. Assuming the ratio 52 : 48, the slag $\frac{CaO,\ 52\%}{Al_2O_3,\ 18\% + SiO_2,\ 30\%}$ would be a good one, while the slag $\frac{CaO,\ 52\%}{Al_2O_3,\ 10\% + SiO_2,\ 38\%}$ would be a bad one, as it would not run well, would

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[1] Ledebur, "Handbuch," 4th ed., p. 592.

[2] Jüptner, The Sulphur Contents of Slags and Other Metallurgical Products: "Journal Iron and Steel Institute," 1902, I, p. 304.

[3] Literature, see p. 21 and .

[4] Platz, "Stahl und Eisen," 1892, p. 2; Nau, "Iron Age," January 28, 1904; Welcke, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1896, p. 135; "Chemiker Zeitung," 1895, p. 2211; Blum, "Stahl und Eisen," 1901, p. 1024; Jüptner, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1902, p. 166.

stick (lime stick). The ranges of SiO_2, Al_2O_3, and CaO for coke and charcoal furnaces were given above.

Let it be supposed, in the calculation of a charge, that the fuel consumption is to be 1 ton of coke to 1 ton of pig, and that the pig is to contain 1 per cent. Si: —

ANALYSES OF ORE, FLUX, AND FUEL.

	Fe	CaO	Al_2O_3	SiO_2	MnO	S
Ore	50	2.7(2)	12	14.16(13)	0.6	0.4
Limestone	. .	51	1	4	. .	. .
Coke	. .	2	3	5	. .	. .

In order to find the amount of SiO_2 that has to be deducted from the iron ore to provide for the Si of the pig, let the pig contain 93 per cent. Fe; then Fe : Si = 93 : 1 = 50 (in ore) : x, $x = 0.54$. This corresponds to 1.16 SiO_2, as Si : SiO_2 = 28 : 60 = 0.54 : y, $y = 1.16$. From the total SiO_2 in the ore (14.16 per cent.) there will have to be deducted 1.16 per cent., leaving 13 per cent. to be slagged. As one-half to four-fifths of the Mn goes into the pig, the small amount of MnO remaining to be slagged can be passed over, otherwise it would have to be considered as replacing an equivalent amount of CaO. The 0.4 per cent. S of the ore has to be combined with Ca to CaS. Now S : CaO = 32 : 56 = 0.4 : x, gives $x = 0.7$. If this be deducted from the 2.7 per cent CaO of the ore, leaving 2 per cent. CaO to be combined with SiO_2 and Al_2O_3, the S will have been provided with the necessary Ca.

Method A. Charge calculated to form a singulosilicate slag. See also Ledebur-Nitze;[1] Surls;[2] Stone;[3] Amsden;[4] Bjerregard;[5] Rossi;[6] Ledebur's "Manual," fourth edition, p. 602. Balling's table[7] on the amounts of SiO_2 the different bases require to form a singulo-, sesqui-, and bi-silicate slag, and conversely the amounts of the different bases required by a unit of SiO_2, greatly facilitates the calculation.

[1] "Engineering and Mining Journal," December 30, 1890.

[2] "Iron Age," February 15, 1891.

[3] *Op. cit.*, March 12, 1891.

[4] *Op. cit.*, January 1, 1892.

[5] *Op. cit.*, October 5, 1893.

[6] *Op. cit.*, April 9, 16, 30, December 3, 1891; February 5, March 3, 1892.

[7] "Metallurgische Chemie," Bonn, 1882, p. 98; or Hofman, "Lead," New York, 1899, p. 275.

TABLE OF BALLING.

One part by weight of silica requires	Parts by weight of bases.	One part by weight of bases requires	Parts by weight of silica.
For singulosilicates: —		For singulosilicates: —	
Lime.	1.86	Lime.	0.535
Baryta.	5.10	Baryta.	0.196
Magnesia.	1.33	Magnesia.	0.750
Alumina.	1.14	Alumina.	0.873
Ferrous oxide.	2.40	Ferrous oxide.	0.416
Manganous oxide.	2.36	Manganous oxide.	0.422
Lead oxide.	7.43	Lead oxide.	0.134
For bisilicates: —		For bisilicates: —	
Lime.	0.93	Lime.	1.070
Baryta.	2.55	Baryta.	0.392
Magnesia.	0.66	Magnesia.	1.500
Alumina.	0.57	Alumina.	1.747
Ferrous oxide.	1.20	Ferrous oxide.	0.883
Manganous oxide.	1.18	Manganous oxide.	0.845
Lead oxide.	3.71	Lead oxide.	0.269
For sesquisilicates: —		For sesquisilicates: —	
Lime.	1.24	Lime.	0.803
Baryta.	3.40	Baryta.	0.294
Magnesia.	0.88	Magnesia.	1.125
Alumina.	0.76	Alumina.	1.310
Ferrous oxide.	1.60	Ferrous oxide.	0.625
Manganous oxide.	1.57	Manganous oxide.	0.633
Lead oxide.	4.95	Lead oxide.	0.202

(1) Available CaO in 100 pounds limestone. 1 pound SiO_2 requires 1.86 pounds CaO, hence 4 pounds SiO_2 require $4 \times 1.86 = 7.44$ pounds CaO. 100 pounds limestone have available $51 - 7.44 = 43.56$ pounds CaO.

(2) Flux required for 100 pounds ore.

$$\text{In 100 pounds ore} \left\{ \begin{array}{lr} \text{2 pounds CaO slag } 2 \times 0.535 = & 1.070 \text{ pounds } SiO_2, \\ \text{12 pounds } Al_2O_3 \text{ slag } 12 \times 0.873 = & \underline{10.476} \text{ pounds } SiO_2, \\ & 11.546 \text{ pounds } SiO_2. \end{array} \right.$$

(2) Flux required for 100 pounds ore: $(Al_2O_3 + SiO_2) : CaO = 48 : 52 = 25 : x, x =$ 27.08 CaO. There are 2 pounds CaO in the ore, leaving 25.08 pounds CaO to be provided for. 100 limestone: 45.58 available CaO $= y : 25.08, y = 55$ pounds limestone.

(3) Flux required for coke. The calculation above showed that 100 pounds ore required 53.8 pounds coke which contained 1.076 pounds CaO, 1.614 pounds Al_2O_3, and 2.690 pounds SiO_2. $(Al_2O_3 + SiO_2) : CaO = 48 : 52 = 4.304 : x, x = 4.662$ pounds CaO. There are present already 1.076 pounds CaO; hence $4.662 - 1.076 = 3.586$ pounds CaO have to be added. 100 limestone: 45.58 available CaO $= y : 3.586, y = 7.86$ pounds limestone.

(4) Summing up.

MATERIAL.		SiO_2, pounds.	Al_2O_3, pounds.	CaO, pounds.	CaS, pounds.
Name.	Pounds.				
Iron ore	100	13.00	12.00	2.00	. . .
Coke	54	2.69	1.61	1.07	. . .
Limestone . . .	63	2.52	0.63	32.13	. . .
Total	217	18.21	14.24	35.20	. . .
Slag	100	26.95	21.07	52.09	. . .
Factor = 1.48		48.02		52.09	. . .

This slag runs rather high in Al_2O_3 and will require more fuel than is desirable, and a highly superheated blast to smelt satisfactorily. In order to obtain a more satisfactory slag, the ore (A) will have to be mixed with a second one (B) running low in Al_2O_3. The proportions will be found by

$$A + B = 100.$$

$$A\,x\,a \text{ per cent.} + B\,x\,b \text{ per cent.} = 100\,x \text{ available per cent.}$$

SECTION 24. **Management of the Blast Furnace.**[1] (1) *Blowing in.* A furnace that has been newly built, or has been relined, must first be dried. The tuyère arches are closed with brickwork. A temporary fireplace is erected at the front and a fire kept going in it for a fortnight or longer. In order to diminish the draught and to force the hot gases to remain longer in the furnace, the top is partly covered over. The furnace is now ready for filling. The lower part, half way up the boshes, is filled with cord wood, then follows a heavy bed of coke, with some limestone to flux the coke ash. This is followed by successive layers of the normal charge of coke, with gradually increasing amounts of ore and limestone and decreasing amounts of slag, until with the last charge the normal amount of ore and flux has been reached. The blowing-in charges are lowered by a rope and carefully spread out. The furnace is lighted

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[1] "School of Mines Quarterly," XIX, p. 227.

at the tuyères (often by hot blast) and allowed to burn. Sometimes a weak blast is used from the first; if not, it is put on later and its volume slowly increased.

(*a*) Blowing in of the Durham Furnace, Riegelsville, Pennsylvania[1] (Figure 122). A cribbing of wood was put into the crucible (11 feet 4 inches diameter) reaching up to the tuyères. On top of cribbing came a layer of cord wood placed horizontally, then three tiers of cord wood stacked vertically, and on top of this a blank charge of 400 cwt. fuel (one-fourth coke, three-fourths anthracite). Then came the charges as follows: —

Charges.	Fuel, cwt.	Ores (52% Fe), cwt.	Limestone, cwt.	Slag, cwt.	Iron in ores, tons.
Bed	400	. . .	20	80	. . .
4	80	30	17	20	3.12
4	80	35	20	17½	3.64
4	80	40	22½	15	4.16
4	80	45	25	12½	4.68
4	80	50	27½	10	5.20
4	80	60	32½	7½	6.24
4	80	70	40	5	7.28
4	80	80	45	2½	8.32
4	80	88	50	. . .	9.15
4	80	96	52½	. . .	9.98
4	80	104	57½	. . .	10.82
4	80	108	60	. . .	11.23
Total tons	212	$161\frac{4}{20}$	$90\frac{18}{20}$	22	83.82

Record of time:[2] fired up March 23, 4.30 P.M.; put on blast March 24, 9.30 P.M.; tapped first cinder March 25, 1.30 P.M.; tapped first iron March 26, 8.30 A.M.

Record of pig made: —

Kind of pig.	Day.							Total.
	First.	Second.	Third.	Fourth.	Fifth.	Sixth.	Seventh.	
Number 2	. . .	7½	. . .	. . .	. . .	. . .	. . .	7½ tons
Gray forge . . .	7½	17½	37½	62½	70	67½	70	332½ tons

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[1] "Transactions American Institute of Mining Engineers," XVIII, p. 379.

[2] Detail, see *l. c.*, p. 888.

[illegible]

(*b*) Blowing in of Number 1 Furnace, Ensley, Alabama.[1] The furnace was dried by one month's firing; the stoves were heated by taking gas from adjoining furnaces (first hot blast 427° C.). The hearth was filled with wood as above, then 220,000 pounds coke, then 78,000 pounds with 12 per cent. limestone, then seven charges (coke, 65,000 pounds; ore, 4,000 pounds; limestone, 980 pounds) were given; the burden was raised every seven charges; the last charges consisted of coke, 6,500 pounds; ore, 8,700 pounds; limestone, 2,000 pounds. Greasy waste was placed in front of each tuyère; 7,000 cubic feet hot blast were turned on; the revolutions of the blowing engine were increased until, after twenty-four hours, 16,000 cubic feet per minute were delivered. The first cinder was tapped fourteen hours after lighting; it was hot, gray, and fluid. The first cast was made sixteen hours after lighting; it contained 17½ tons numbers 1 and 2 soft iron. The first complete furnace day gave 84 tons pig (80 per cent. foundry); the third day 142 tons; the sixth day 155 tons (70 per cent. foundry), the charge consisting of coke, 6,500 pounds; ore, 11,680 pounds; limestone, 2,700 pounds.

(*c*) Blowing in of Isabella Furnace, Number 3, Pittsburgh, Pennsylvania, February 14, 1902.

Filling Record. First filling: charge bottom of furnace to level of cinder notch with coke. Second filling: give 2 feet of pine wood. Third filling: 8 feet of oak cord wood. Fourth filling: 47 skips of coke (@ 3,300 pounds); 8 barrows of slag (@ 800 pounds). Fifth filling: 12 skips of coke (@ 3,300 pounds); 6 barrows of slag (@ 800 pounds). Sixth filling: 12 skips of coke (@ 3,300 pounds); 6 barrows of slag (@ 800 pounds each); 4 barrows of limestone (@ 725 pounds). Seventh filling: 12 skips of coke ((@ 3,300 pounds); 4 barrows of slag ((@ 800 pounds); 4 barrows of limestone (@ 725 pounds). Eighth filling: 7 skips of coke (@ 3,300 pounds); 2 barrows of slag (@ 800 pounds); 4 barrows of limestone ((@ 725 pounds). Ninth filling: 6,600 pounds ore, Aurora; 2,200 pounds ore, Norden; 600 pounds ore, Clifford; 3,760 pounds limestone; 9,900 pounds Connellsville coke. Tenth filling: for ten charges, 6,930 pounds ore, Aurora; 2,310 pounds ore, Norden; 630 pounds ore, Clifford; 3,840 pounds limestone (furnace full after ninth charge); 9,900 pounds Connellsville coke. Eleventh filling: for ten charges, 7,280 pounds ore, Aurora; 2,430 pounds ore, Norden; 660 pounds ore, Clifford; 3,920 pounds limestone; 9,900 pounds Connellsville coke. Twelfth filling: for ten charges, 7,730 pounds ore, Aurora; 2,580 pounds ore, Norden; 700 pounds ore, Clifford; 4,020 pounds limestone; 9,900 pounds Connellsville coke. Thirteenth filling: for ten charges, 8,200 pounds ore, Aurora; 2,730 pounds ore, Norden; 600 pounds ore, Clifford; 3,980 pounds limestone; 9,900 pounds Connellsville coke. Fourteenth filling: for ten charges, 8,700 pounds ore, Aurora; 2,900 pounds ore, Norden; 500 pounds ore, Clifford; 3,950 pounds limestone; 9,900 pounds Connellsville coke. Fifteenth filling: 9,200 pounds ore, Aurora; 3,070 pounds ore, Norden; 400 pounds ore, Clifford; 3,920 pounds limestone; 9,900 pounds Connellsville coke. Sixteenth filling: for ten charges, 6,600 pounds ore, Aurora; 2,200 pounds ore, Norden; 3,300 pounds ore, Group Number 1; 1,100 pounds ore, Ajax; 3,700 pounds limestone; 9,900 pounds Connellsville coke.

Blast Record. Blast on at 11.52 P.M., February 14, blowing engine (84-inch x 60-inch cylinder) making fifteen revolutions per minute; relief valve on bustle pipe and bleeder are open; damper valves are closed; there is no gas in the flues. At 7 A.M., February 15, the relief valves are closed (wood smoke having disappeared and good clear coke being visible at the tuyères). Two hours after closing the relief valves the speed of the engine was increased two revolutions per hour until fifty revolutions was reached, and then one revolution per hour until seventy revolutions was reached. The stoves were cold at start (about 26° C.). Gas was turned into the flues at 2.20 P.M., May 15.

Tapping Record. The first cinder was flushed February 16 at 4.30 A.M.; the first iron tapped at 10.30 A.M. (one and one-half ladlefuls).

Analysis of first iron: Si, 2.00 per cent.; S, 0.186 per cent.

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[1] "Proceedings Alabama Industrial and Scientific Society," II, p. 75; "Engineering and Mining Journal," October 22, 1892.

ANALYSES OF ORE, LIMESTONE, AND COKE.

	SiO_2	Fe	P	CaO
Ore, Aurora	4.3	54.6	.037	. . .
Ore, Norden	2.9	57.8	.072	. . .
Ore, Clifford	36.8	38.8	.016	. . .
Ore, Ajax	10.2	51.3	.054	. . .
Ore, Mesabi	3.6	56.3	.086	. . .
Limestone	4.0	. . .	.008	50.00
Coke	6.0	. . .	.015	. . .

Earlier Records: "Transactions American Institute of Mining Engineers," VIII, p. 348 (Kennedy); XVII, p. 138 (Gordon); XX, p. 262 (Gayley).

(2) *Regular Running.* Ore, flux, and fuel are brought in buggies from the stock yard or stock house to the scales, weighed, hoisted to the top of the furnace, and dumped automatically into the bell or wheeled by the top-fillers to the bell and then dumped.[1] The fuel is dropped by itself, ore and flux together. The level of the charge is kept constant, its depth being determined by lowering an iron rod through a hole in the bell.[2] The temperature and pressure of the blast are kept constant; if the temperature becomes too high it is lowered by taking through a by-pass some air from the cold-blast main. The volume of air is kept the same by regulating the revolutions of the engine; a rise in the pressure of the blast indicates trouble. The cooling water is kept constantly running; slag is tapped at intervals (say every two hours), as well as pig (every four to six hours). If the metal flows from the runner into molds, the pigs are severed from the sow while still hot by sledges and stacked in the yard when cold. Here they are broken by dropping on to a heavy casting of the form of an inverted V, or by a pig-breaking machine,[3] and assorted according to fracture. If the metal is collected in ladles a switching engine takes them to the mixer or to a casting machine.

(3) *Banking the Furnace.* When a furnace has to be temporarily shut down on account of a falling market, a flood, a strike, or some other cause, it is banked. A large blank charge of coke, with the necessary limestone, is given. This is followed by blowing in charges, *i.e.*, by one-half, two-thirds, three-fourths, and then again by full ore charges. The furnace is run until the blank arrives at the tuyères, when the blast is shut off, the tuyères are removed and their openings bricked up, the bell sealed with fine ore, and the cooling water gradually decreased. If the furnace has to be banked suddenly (as in case of a strike), all slag and metal are tapped and all openings closed as before. If the banking can be planned for, the furnace can remain out of blast two to three months; if not, only two to three weeks.

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[1] Baker, Handbarrow and Automatic Filling: "Transactions American Institute of Mining Engineers," February, 1904; Barrows and Uehling, *op. cit.*, February, 1904.

[2] Baker, Stock Indicator: "Iron Age," July 18, 1901.

[3] "Transactions American Institute of Mining Engineers," XXVII, p. 38; "Iron Age," July 6, 1893; August 2, 1894; "Berg- und Hüttenmännische Zeitung," 1897, 1898; "Stahl und Eisen," 1902, p. 150.

(4) *Blowing Down.* When a furnace is to be put out of blast, the charges are stopped and a layer of coke of sufficient depth to maintain a reducing atmosphere is given with the necessary flux to slag the coke ash. Tapping of iron and slag is kept up as long as anything will come, the blast being diminished in volume as the charges descend. In order to keep the furnace cool, a jet of water is run in through a hole in the bell or in the charging floor.[1] A method common in some European plants since the sixties is to charge limestone in sufficient quantity to keep down the flame, the resulting burnt lime being used for mortar. The Cedar Point Furnace, Port Henry, New York,[2] was thus filled with 108 tons limestone, the bell sealed, filled with H_2O, and kept filled until the blowing out was finished, the pressure of the blast being kept high, which made it possible to burn out all the coke. When the blast was taken off, the furnace contained only lime above the tuyères and an insignificant salamander below. The lime was washed out fifteen hours after the blast was stopped.

(5) *Irregularities in the Furnace.*[3] The main principle to be followed in furnacing is to keep the top cool and the bottom hot (cool head, warm feet). Irregularities are caused by a chill, the hanging and irregular descent of the charges, or faulty fluxing. They are indicated: —

By the slag becoming green, then black, from slagged iron, and full of shots of iron (buckshot), or by its not flowing on account of the excess of a constituent; by the pig changing from gray to white iron.

By insufficiently carburized iron (wrought iron) reaching the crucible, forming a sow.

By cyanonitride of titanium (from titanic ores) settling in the crucible, etc.

A chill is caused by too heavy a burden, by moist charges, by sudden change of moisture in atmosphere (on the average 125 pounds of water, in moist summer weather even 200 pounds water, are forced into the furnace in the form of vapor per ton of iron); by inferior fuel; by a low temperature — insufficient volume and pressure of blast; by leakage of water; by irregular descent of charge, causing Fe_2O_3 to be reduced by C instead of by CO.

The hanging,[4] *i.e.*, the not descending of charges which form scaffolds[5] (hangers) and arches a short distance above the tuyère and at the bosh, is caused by the fusion beginning too high up in the furnace, on account of a too fusible slag, and causing the formation of accretions. If they form on one side (hangers, scaffolds) they force the charges to descend entirely on the opposite side, and interrupt the even ascent of the gases. If they form all around they cause the charge to arch over without descending at all. The charges below the arch will be smelted away, leaving a hollow space. Hanging and choking of the furnace (explosions) are caused near the top by the excessive carbon deposition ($2CO = C + CO_2$) from finely divided, readily reducible hematite ores.[6] If a chill should be caused by too quick a descent of charges, when reduction will take place by C instead of by CO, the temperature of the blast will have to be lowered; as a rule, however, the cause of the chill will be a lack of heat. This can be remedied by raising the temperature or by lightening the burden. The second way works slowly, only producing

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[1] "Transactions American Institute of Mining Engineers," XIII, p. 670.

[2] *Op. cit.*, VI, p. 169.

[3] Walsh, *op cit.*, XV, p. 419.

[4] "Stahl und Eisen," 1892, pp. 114, 336, 467, 528, 582; 1901, p. 1277; 1902, p. 258; 1903, pp. 773, 838.

[5] "Transactions American Institute of Mining Engineers," IX, pp. 41, 60; XIII, p. 675.

[6] **Explosions:** "Stahl und Eisen," 1903, pp. 623, 922; "Journal Iron and Steel Institute," 1903, II, p. 617; "Iron Age," July 2, 1903; April 7, 1904.

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ASTOR, LENOX AND
TILDEN FOUNDATIONS.

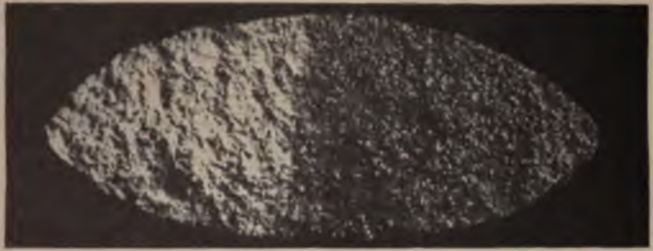

Fig. 125. Fracture of Gray Iron, Graphite Brushed Away from One-half.

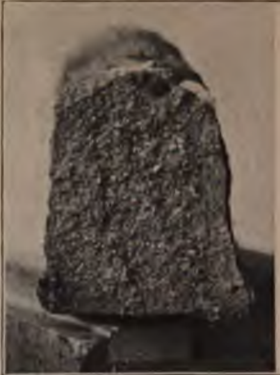

Fig. 124. Gray Iron.

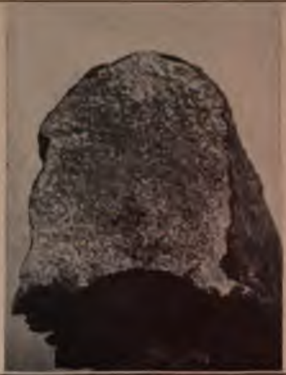

Fig. 128. Mottled Iron.

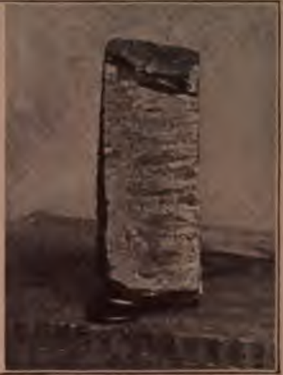

Fig. 127. White Iron.

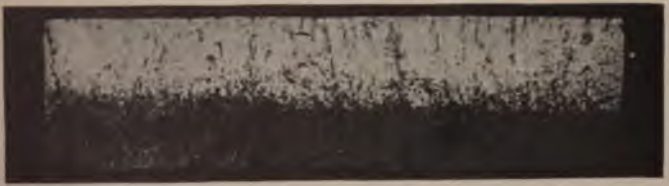

Fig. 130. Chill Test Piece.

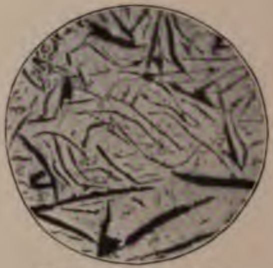

Fig 123. Gray Iron.

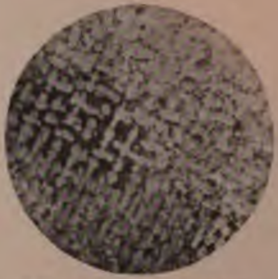

Fig. 129. Mottled Iron.

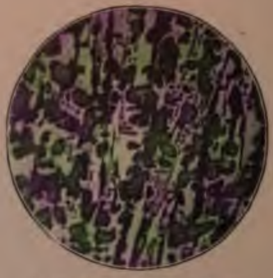

Fig 126. White Iron.

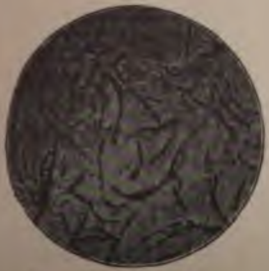

Fig. 132. Coke Iron.
(Phœnix.)

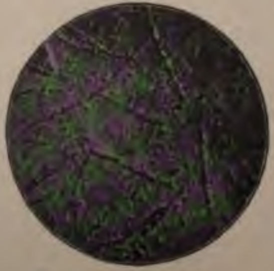

Fig. 133. Coke Iron.
(Buderus.)

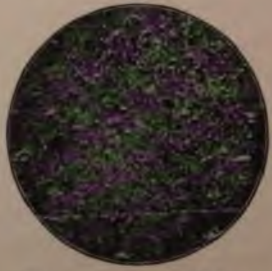

Fig. 134 Charcoal Iron,
Cast in Dry Sand.

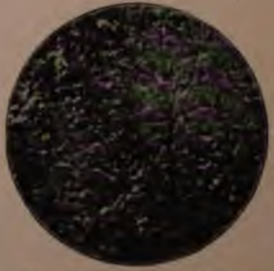

Fig. 135. Charcoal Iron,
Cast in Moist Sand.

an effect after twenty to forty-eight hours, the time it takes a charge to pass through the furnace. The hanging charges often drop (sometimes accompanied by explosion), the material below them having given way. This dropping may be helped by slackening the blast. The speed of the engine may be lowered and thus the temperature of the blast raised, when the scaffold or arch may be melted away. Scaffolds are sometimes removed by "fighting" with a bosh tuyère (36, Figure 36) with explosives, etc.[1] Hangings from carbon deposition may be corrected by lowering the blast temperature, when with a larger volume of air the heat will creep up and the C be burned away. On the other hand, their formation can be prevented by raising the temperature above 685° C. (see p. 68). Accretions in the hearth may be removed by a petroleum blowpipe,[2] or by explosives,[3] or by an electric burner.[4] Wrought iron and titanic salamanders are removed after the furnace has been blown down. They often extend several feet into the ground, and have to be blasted with dynamite and lifted out by a derrick.

SECTION 25. **Products of the Blast Furnace.** The main products of the blast furnace are pig iron, slag, and waste gases; and the incidental products, flue dust, cadmies, sows, lead, etc.

SECTION 26. **Pig Iron.**[5] Cast iron is classified (*a*) according to the general appearance of the fracture and the chemical composition as gray iron (including silicon irons) and white iron (including the iron-manganese series), each of which has many subdivisions; (*b*) according to the fuel with which it was made, as coke, anthracite, and charcoal iron; (*c*) according to the use it is put to, as acid and basic Bessemer iron, acid and basic open-hearth iron, foundry iron, mill iron, and special irons, such as the ferrosilicon and ferromanganese series, used in small quantities in Bessemer, open-hearth, and foundry work.

(1) *Gray and White Iron.* The characteristic of gray iron is the presence of graphite and silicon; the former, separating out at the moment of solidification, gives it its gray color. An etched, polished surface (Figure 123) will show under the microscope grayish graphite, lighter ferrite, and small amounts of cementite and pearlite; the fracture when not polished (Figure 124) shows scaly graphite. The loose condition of the graphite disseminated through the iron matrix is shown in Figure 125.[6] It represents the fracture of an arm of a pulley cast from soft iron; one-half appears almost white, the other dark gray. The change in appearance was effected by brushing one-half of the fracture briskly with a whisk for three minutes, whereby the loose graphite lying between the crystals was brushed away like dust. In white iron graphite is absent, and Si present only in very small amounts, if at all. The polished etched surface (Figure 126) shows white cementite and dark pearlite (martensite); the fracture (Figure 127) is finely granular, often radiated, the color white. Between these two extreme forms there exists a number of gradations, forming the subdivisions given below. Figure 128 shows a fracture, Figure 129 a photomicrograph of mottled iron. As slow cooling favors the separation of graphite while quick cooling (chilling) impedes it, the same iron may be white on the surface

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[1] "Transactions American Institute of Mining Engineers," XIII, p. 670.

[2] *Op. cit.*, XV, p. 47.

[3] *Op. cit.*, X, p. 206.

[4] *Op. cit.*, XXXI, p. 626.

[5] Ledebur, "Das Roheisen," 4th ed., Felix, Leipsic, 1904; Meissner, "School of Mines Quarterly," X, p. 147; W. J. Keep, "Cast Iron," New York, 1902; S. R. Church, "Analyses of Pig Iron," 2 vols., San Francisco, 1900 and 1901.

[6] Outerbridge, "Journal Franklin Institute," 1900, CL, p. 341, Figure 10.

and gray on the inside; provided, of course, that the elements, such as Si, Mn, and S, are not present in extreme quantities to prevent the change. The change in appearance of fracture is strikingly shown in Figure 130,[1] which represents a chill test piece that was cast in a green sand mold having an iron chill block on one face. The part in contact with the sand, cooling slowly, gave the fracture of gray iron; that in contact with the iron, being chilled, the fracture of white iron. Differences of fracture of machine and sand-cast pig iron have been clearly illustrated by Colby.[2]

The separation of graphite upon cooling causes gray iron to increase in volume, hence its specific gravity is lower (7.0–7.2) than that of white iron (7.6). The melting-point[3] of gray iron is higher[4] (1,220° C.) than that of white iron[5] (1,065° C.), as gray iron, consisting mainly of iron (low in or free from combined carbon) and graphite, has to be heated until the graphite is dissolved and combined before it can melt; while in white iron the carbon is already present (mainly) in the combined state. In fact, the melting-point rises as the combined carbon decreases. Gray iron liquefies suddenly upon heating; white iron softens before melting. The fluidity of the iron is greatly influenced by the other elements beside C; thus Si, Mn, and especially P increase it; small amounts of S and any oxidized iron decrease it.

(2) *Carbon.*[6] Gray iron is soft on account of the predominance of graphite and ferrite, and is readily machined; white iron is hard on account of the prevalence of cementite, and can be filed and cut only if the amount of C does not exceed 3 per cent., and then very hard steel tools must be used. Cast iron is brittle; its tensile strength and compressive strength vary with the structure and composition. The coarser grained the structure the weaker the iron; hardening carbon in small amounts causes a finer granulation, and therefore makes the iron strong; large quantities, however, make the iron brittle, and therefore weak. As gray iron contains but small quantities of hardening C it is stronger than white iron.- Gray iron running very high in graphite is weaker than when small amounts of C. C. are present. A strong foundry iron should contain not over 3 per cent. Gr. C., with 3.5 per cent. T. C., and not less than 2 per cent. Gr. C. with 2.6 per cent. T. C. The necessity for having definite proportions of combined and graphitic carbon in pig iron of a given carbon content, if the greatest strength is to be obtained, has been explained by Howe[7] by his theory of the constitution of cast iron. He says that cast iron is made up of two distinct parts: first, graphite, which, breaking the continuity, weakens the iron; and second, metallic matrix, identical in its properties with slowly cooled steel, which gives the iron its strength. In Figure 131 it is shown (as has been shown already in part in Figure 24) that the tenacity of steel reaches a maximum at about 1.25 per cent. C and then decreases. From Figure 18 (page 11) it is seen that such a steel is composed of structurally free cementite and pearlite (88 per cent. ferrite + 12 per cent. cementite).

[1] Outerbridge, "Journal Franklin Institute," 1900, CL, p. 340, Figure 5.

[2] "Iron Age," June 20, 1901.

[3] See also Moldenke, "Foundry," XIII, p. 102.

[4] LeChatelier, "Comptes Rendus," CXL, p. 471.

[5] Royston, "Journal Iron and Steel Institute," 1897, I, p. 167.

[6] Scott, "Proceedings American Society for Testing Materials," II, p. 196; "Journal American Foundrymen's Association," XII, p. 323.

[7] "Transactions American Institute of Mining Engineers," XXI, pp. 318, 985; "Proceedings American Society for Testing Materials," II, p. 246.

As the difference between total carbon and combined carbon is graphite, and as this has a weakening effect, it is justifiable to say that the smaller the amount of graphite the stronger is a gray cast iron. But strength is not all. Figure 131 shows that the hardness (and with it the brittleness) increases with the percentage of combined carbon, and in Figure 18 it is seen that this is due to the increase in the amount of glass-hard cementite. The increase of ductility (Figure 131) with the decrease of the carbon content is shown in Figure 18 to be due to the greater amount of soft, malleable ferrite formed with the pearlite, and this makes the iron tough. Thus with different percentages of total carbon there is a best proportion of graphitic and combined carbon. The various changes in the structure of cast iron have been studied micrographically by Campbell.[1]

(3) *Silicon.*[2] Silicon must be present if C is to separate as graphite, because as Si increases the combined carbon decreases.[3] This decrease is rapid at first, especially as the Si increases from 0 to 0.75 per cent., and then slower and slower. Strong and tough cast iron should not contain less than 1 per cent. or more than 4 per cent. Si.[4] Below 1 per cent. Si the iron begins to harden and to produce a chill; above 4 per cent. it begins again to harden and become short and weak (rotten-shortness). The chill increases as the Si decreases; thus cast iron with 1 per cent. Si gives a chill of $\frac{1}{8}$ inch; with 0.70 per cent. Si one of $\frac{1}{4}$ inch; 0.52 per cent. Si, $\frac{5}{8}$ inch; 0.40 per cent. Si, 1 inch; 0.30 per cent. Si, $1\frac{1}{2}$ inches. Good machinery castings of close structure should contain 1.5 to 2 per cent. or even 2.5 per cent. Si.[5] Soft irons show little, hard irons much shrinkage. An increase of 0.20 per cent. Si is equivalent to a decrease in shrinkage of about 0.01 inch per foot. High and low Si impart less strength than intermediate amounts (see table, next page). Shrinkage and hardness are nearly proportionate. Sulphur[6] destroys the effect of Si as a hardener. Thus 0.01 per cent. S will increase the combined carbon to 0.02–0.03 per cent. According to Scott[7] 0.01 per cent. S is equivalent to 0.15 per cent. Si. Si[8] is unevenly distributed in the pigs from a cast, the iron coming first from the furnace having the highest percentage. Lebeau[9] gives reasons for believing that in cast iron Si is present as $SiFe_2$. Al[10] acts in a manner similar to Si, only more energetically, 0.1 per cent. Al, or even less, remaining in the iron. This is seen again by the work of Medland and Waldron,[11] which shows that slow or quick cooling makes little difference in the amount of graphite separating out when Al is present. Si, then, by promoting the formation of graphite, softens iron, imparts fluidity, decreases shrinkage, and opens the grain and reduces the strength.

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[1] "Proceedings American Society for Testing Materials," III, p. 175.

[2] Keep, "Transactions American Institute of Mining Engineers," XVII, p. 683; Scott, "Proceedings American Society for Testing Materials," II, p. 183.

[3] Howe, "Transactions American Institute of Mining Engineers," XXX, p. 731.

[4] Diagram, Turner, "Iron," p. 194.

[5] Cook, "Transactions American Institute of Mining Engineers," XXVIII, p. 397.

[6] Howe, *op. cit.*, XXX, p. 732.

[7] "Proceedings American Society for Testing Materials," II, p. 184.

[8] Thomas, "Iron Age," November 12, 1891.

[9] "Stahl und Eisen," 1902, p. 446.

[10] Keep, "Journal Franklin Institute," CVI, p. 220; "Transactions American Institute of Mining Engineers," XVIII, pp. 102, 798; Borsig, "Stahl und Eisen," 1894, p. 6; Hogg, "Journal Iron and Steel Institute," 1894, II, p. 104.

[11] "Journal Iron and Steel Institute," 1900, II, p. 244; "Stahl und Eisen," 1901, p. 54 (Ledebur).

Si per cent.	Transverse strength.	Tensile strength.	Hardness of iron.
0.20	2,600 pounds	20,500 pounds	Extremely hard.
0.50	2,800 pounds	24,500 pounds	Very hard.
1.00	3,000 pounds	25,500 pounds	Hard.
1.40	3,300 pounds	26,500 pounds	Medium hard.
1.90	2,900 pounds	31,500 pounds	Medium soft.
2.20	2,500 pounds	30,000 pounds	Soft.
2.50	2,300 pounds	29,000 pounds	Very soft.
3.00	2,000 pounds	24,500 pounds	Extremely soft.
3.50	1,700 pounds	22,500 pounds	Slightly hard.

(4) *Manganese.* Manganese,[1] if under 1.0 per cent., strengthens cast iron; 1.0–1.5 per cent. strengthens it but makes it brittle; if over 1.5 per cent. it decreases the strength and toughness and increases hardness and shrinkage greatly. If in ordinary cast iron (Si, 1.80–2.50 per cent.) Mn sinks below 0.25 per cent., there is liable to be trouble from blowholes, dirt, etc. Ordinary castings contain about 0.40 per cent. Mn. Large castings, remaining fluid a long time and cooling slowly, may contain as much as 1 per cent.; medium size, 0.60 per cent.; small, 0.30 per cent. With high Si (2.30–2.80 per cent), large castings may contain 1.25 per cent. Mn; small castings, 0.60 per cent., without becoming hard.

(5) *Phosphorus.*[2] Phosphorus, even in small quantities, makes the iron weak and brittle. Above 0.8 per cent. P, cast iron becomes cold-short; above 1.20 per cent., graphitic carbon changes to combined carbon; above 1.60 per cent., cast iron becomes hard-short (a slight shock breaks castings). The more hardening carbon and manganese the iron contains the more detrimental is the effect of P. Thus in chilled castings (C. C., 1.00–1.50 per cent.) P ought to be below 0.3 per cent.; in malleable castings (which are low in Si) it ought to be below 0.20 per cent., while in gray castings (high in Si) it can reach 0.7 per cent. P slightly increases the fluidity and makes iron retain its heat longer. Shrinkage decreases with increase of P; 0.01 per cent. P is about equivalent to 0.03 per cent. Si. Gray iron to be used for machinery castings should not contain over 1 per cent. P, and is better kept at 0.5 per cent.; if used for other purposes the P may reach 1.5 per cent., *e.g.*, with stoves.

(6) *S, As, Sb, Cu, Ti, etc.* The effect of these impurities on gray iron is to weaken it and to increase its brittleness.

Sulphur makes iron run sluggishly and gives rise to blowholes. This is counteracted by Mn; with Mn 0.30 per cent., the S should not exceed 0.065 per cent.; with Mn 0.60 per cent., S may rise to 0.09 per cent.; with Mn 0.75 per cent., S may reach 0.130 per cent. and give a fair casting. Mn increases the shrinkage and chill of Fe. Every 0.03 per cent. increase of S corresponds to 0.01 inch increase in shrinkage per foot. Good foundry, *i.e.*, low-P iron should

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[1] Keep, "Transactions American Institute of Mining Engineers," XX, p. 291; West, "Iron Trade Review," July 11, 1901.

[2] Keep, "Transactions American Institute of Mining Engineers," XVIII, p. 458.

not contain over 0.15 per cent. S, while with medium- to high-P iron, 0.08 per cent. is a safe limit.

Little is known about the effects of As and Sb; they are present rarely to a greater amount than 0.1 per cent. Cu is present in cast iron in very small quantities only. According to Lipin[1] cast iron can take up 5 per cent. Cu, while Stead[2] found 7.5 per cent. to be the limit. Cu increases the fluidity, makes the casting denser, and increases the tenacity. Ti[3] enters to a small extent gray but not white iron, giving it greater tenacity; by reacting upon O and N held in solution an addition of 2–3 per cent. of 10 per cent. titanium alloy increases the strength of cast iron by 20 per cent. Blair-Shimer[4] isolated a titanium sulphide from cast iron.

(7) *Gases in Cast Iron.* Münker, "Stahl und Eisen," 1904, p. 23.

(8) Tensile strength of common iron is 14,000–20,000 pounds (Lanza), 13,400–29,000 pounds, average 16,500 (Rankin). Tensile strength of gun iron is 30,000–33,000 pounds (Lanza). Compressive strength is 82,000–145,000, average 112,000 pounds (Rankin). Modulus of elasticity, 14,000,000–22,900,000, average 17,000,000 pounds (Rankin). Outerbridge[5] found that, within limits, cast iron is materially strengthened by being subjected to repeated shocks and blows, instead of becoming weakened, as has been usually accepted.

SECTION 27. **The Grading of Cast Iron.**[6] (1) *Ferrosilicon* (silvery iron) contains 5–16 per cent. Si, 3–1 per cent. C (present almost exclusively as graphitic carbon, the C decreasing with an increase of Si); it runs low in Mn, and as a rule also in P; it is close-grained (the closer the higher the Si); silvery (the lighter the higher the Si); flaky; shows little or no granulation; is weak.

Locality.	Si	T. C.	Gr. C.	Mn	P	S
Hörde[1]	16.31	0.80	0.80	1.22	0.18	n. d.
Pencost[2]	12.08	1.58	1.52	0.76	0.48	Trace
Pencost[2]	10.34	1.99	1.92	0.52	0.45	Trace
Globe[2]	5.89	3.15	2.85	1.00	1.10	0.02
Brand, Ashland[3]	12.24	1.23	1.10	0.41	0.093	. . .
Brand, Ashland[3]	12.83	1.48	1.36	0.17	0.105	0.05
Brand, Star[3]	7.97	2.02	1.84	0.12	0.92	0.008
Brand, Star[3]	9.33	2.22	2.13	0.42	0.96	0.009

[1] Ledebur, "Manual," 4th ed., p. 401.
[2] "Transactions American Institute of Mining Engineers," XVII, p. 255.
[3] *Op. cit.*, XXVIII, p. 783.
See also Turner, "Iron," p. 214. Ca and Mg have been found by Gray ("Journal Iron and Steel Institute," 1901, II, p. 144) and Ledebur ("Stahl und Eisen," 1902, p. 710).

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[1] "Stahl und Eisen," 1900, p. 537.

[2] "Journal Iron and Steel Institute," 1901, II, p. 114.

[3] Rossi, "Journal American Foundrymen's Association," 1901, X^1, p. 45; "Transactions American Society Mechanical Engineers," 1901, XXII, p. 570; "Iron Age," February 20, 1896, p. 467.

[4] "Transactions American Institute of Mining Engineers," XXXI, p. 748.

[5] *Op. cit.*, 1896, XXVI, pp. 176, 1019, 1023; February, 1904.

[6] Classification according to fracture and chemical composition.

In making ferrosilicons in the blast furnace, the ore ought to be readily reducible that the Fe_xO_y may be totally reduced to Fe by CO; the burden light; the slag rich in Al_2O_3; and the blast hot, so as to have the high temperature in the smelting zone requisite for the reduction of SiO_2 by solid C. The iron flows quickly without sparking.

SLAG ANALYSES.

SiO_2	Al_2O_3	FeO	MnO	CaS	CaO	MgO	Oa : Ob
36.6*	26.0	. . .	1.0	. . .	33.0	. . .	0.8
33.10†	24.56	0.31	0.37	7.76	25.92	6.97	0.791
27.70†	26.70	Trace	Trace	5.43	34.69	3.45	0.462

* "Oesterreichisches Jahrbuch," 1896, p. 135. † Ledebur, "Manual," 4th ed., p. 592.

Ferrosilicon is used as a softener for iron castings when high in P, and as a deoxidizer for steel castings when low in P.

(2) *Gray Irons.*[1] (*a*) Numbers 1 and 2 Soft Irons, Si, 3.5–5, C, 3.5–2.5 (present mostly as Gr. C.), begin to show granulation, have a light color, a smooth surface, and a weak fraction; are less flaky than ferrosilicons, less jagged than foundry irons, and darker than ferrosilicons.

Si average.	Si	T. C.	Gr. C.	Mn	P	S	
4–4.25	4.10	3.100	2.635	0.37	1.07	0.099	Buffalo
3–3.25	3.14	3.775	3.574	0.68	0.39	0.023	
. . .	3.00	3.43	3.29	0.24	0.84	0.013	Alabama

REFERENCE. — "Transactions American Institute of Mining Engineers," XXVIII, pp, 796, 784.

In smelting for gray irons the temperature has to be lower than for ferrosilicon, hence the burden can be heavier and the formation temperature of the slag lower. The general run of slags for charcoal and coke furnaces has been given on page 77.

(*b*) Foundry Irons. Number 1 Foundry Iron, Si, 2.50–3.50, is dark, has a granulation that is very open and regular; its crystallization is well marked, and the face is rough to the feel.

Number 2 Foundry Iron, Si, 2.25–2.50, has the same general character as Number 1. It is closer grained, shows a finer crystallization, and while the color may be the same as in Number 1, it lacks the blackish-gray, velvety quality.

Number 3 Foundry Iron, with Si, 2.00–2.25 per cent., and Number 4 Foundry Iron, with 1.75–2.00 per cent. Si, form the transition to gray forge.

The total carbon of Numbers 1 and 2 varies from 4–3.5 per cent.; in Numbers 3 and 4 it rarely exceeds 3.5 per cent., and the graphitic carbon of Numbers 3 and 4 is about 3 per cent. These irons flow sluggishly in the run.

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[1] Phillips, "Journal American Foundrymen's Association," VI, p. 6.

Si average.	Si	T. C.	Gr. C.	Mn	P	S
3.25–3.50	3.32	3.235	2.802	0.37	0.89	0.112
3.00–3.25	3.08	3.4456	3.097	0.50	0.75	0.071
2.75–3.00	2.86	3.256	2.91	0.47	0.71	0.074
2.50–2.75	2.61	3.243	2.856	0.41	0.75	0.090
2.25–2.50	2.37	3.390	2.9366	0.48	0.79	0.073
2.00–2.25	2.12	3.3483	2.9125	0.43	0.75	0.068
1.75–2.00	1.90	3.423	2.935	0.38	0.79	0.068

REFERENCE. — "Transactions American Institute of Mining Engineers," XXVIII, p. 796.

(*c*) Gray Forge Iron, Si 1.50–1.75 per cent., has a pebbled granulation, the face is blistered, pitted, often honeycombed. It flows less sluggishly than foundry irons and sparks.

(*d*) Mottled Iron, Si 1.25–1.50 per cent., C 3–3.5 per cent., Gr. C. 2 per cent. or less, has next to no granulation, but shows a pepper-and-salt appearance. It flows readily and sparks; is formed, as a rule, when the blast furnace is out of order. Gray forge and mottled iron are used both in the foundry and in the rolling mill.

Si average.	Si	T. C.	Gr. C.	Mn	P	S
1.50–1.75	1.62	3.4627	2.959	0.42	0.67	0.08
1.25–1.50	1.38	3.322	2.773	0.45	0.59	0.10

REFERENCE. — "Transactions American Institute of Mining Engineers," XXVIII, p. 796.

(3) ***White Iron***, Si, 0.1–0.6, C, 3–4.5, all combined, has no granulation whatever; is white but not silvery; flows slowly; sparks; is used in the rolling mill exclusively.

Si	T. C.	Gr. C.	Mn	P	S
0.53 *	3.18	. . .	0.78	0.96	0.07
0.41 †	3.83	. . .	0.98	0.04	0.02
0.28 *	3.03	. . .	0.16	0.02	0.10

* Ledebur, "Manual," 4th ed., pp. 407, 596.

† "Journal Franklin Institute," CXXXIV, p. 132.

In smelting for white iron a reduction of SiO_2 has to be prevented, hence a heavy burden and readily fusible slag are required. Ores rich in Mn are smelted for white iron.

Slags.	SiO_2	Al_2O_3	Bases.
Charcoal	45–50	10–5	45–45
Coke	30–40	10–5	60–55

REFERENCE. — Ledebur, "Manual," 4th ed., p. 599.

Mn favors the formation of white iron; P is present in varying quantities, from under 1 per cent. to 3 per cent. and over; S is liable to be found in larger amounts than in gray irons, as white irons are produced at temperatures so low that a slag rich enough in CaO to carry off the S would hardly melt. As, however, Mn eliminates the S, sulphurous ores can be smelted in the presence of Mn and a pig low in S produced.

Phillips, arguing that the grading by silicon content alone is unsatisfactory, says that the ratio between Si and C. C. ought to be added for soft and foundry irons, and the percentage of S for gray forge, mottled, and white irons. He proposes the following six grades instead of the original thirteen: —

Grade.	Si	C. C.	S
Silvery irons	5–6	0 10–0.30	0.01–0.54
Soft irons	3–5	0.20–0.60	0.01–0.35
Foundry irons	2–3	0.30–0.90	0.01–0.57
Gray forge	1–2	0.40–1.25	0.04–0.09
Mottled	0.6–1.0	0.50–1.80	0.06–0.11
White	0.1–0.6	1.00–2.50	0.08–0.30

GRADING IN ALABAMA.[1]

Grade.	Gr. C.	C. C.	Si	S	Mn	P
Silver gray	3.13	0.02	5.5	Trace	0.25	0.68
Number 2 soft	3.48	0.03	3.5	0.004	0.26	0.68
Number 1 soft	3.53	0.03	3.5–4.0	0.005	0.27	0.68
Number 1 foundry	3.49	0.07	2.8–3.5	0.005	0.25	0.68
Number 2 foundry	3.55	0.07	2.2–2.6	0.024	0.22	0.68
Number 3 foundry	3.48	0.10	2.0–2.4	0.025	0.21	0.64
Gray forge	3.00	0.57	1.3–1.7	0.045–0.07	0.19	0.64
Mottled	2.11	1.22	1.1–1.6	0.125	0.14	0.64
White	0.10	2.92	0.7–1.2	0.300	0.10	0.64

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[1] "Iron Age," June 4, 1891, p. 1058; "Journal Iron and Steel Institute," 1891, II, p. 245.

Cook[1] in 1899 suggested the following six grades for foundry irons: —

Grade.	Number 1.	Number 2 x.	Number 2 soft.	Number 2 plain.	Number 2 strong.	Number 2 foundry.
Si	2.50–3.00	2.00–2.50	2.50–3.00	2.00–2.50	1.50–2.00	1.50–2.50
S	0.02 and less	0.02 and less	0.5 and less	0.5 and less	0.5 and less	over 0.5

He simplified this in 1904,[2] as seen in the subjoined table: —

Grade.	Number 1.	Number 2.	Number 3.	Number 4.
Si	2.75	2.25	1.75	1.25
S	0.035	0.045	0.055	0.065

(4) *Spiegeleisen, Ferromanganese, and Siliconspiegel.* Spiegeleisen is a white iron with 4–5 per cent. C, 5–20 per cent. Mn, little S, and not more than 0.15 per cent. P, as on account of its being used as a deoxidizing and recarburizing agent with steel the P must be low. The fracture shows large reflecting surfaces.

Ferromanganese is a white, granular iron with 25–85 per cent. Mn, running high in C, as the percentage of C increases with the Mn. The original white color soon becomes yellowish; cracks form on cooling, showing iridescent colors.

Siliconspiegel is a spiegeleisen with as much as 12 per cent. Si; the percentage of C has been lowered by the Si, but remains mostly combined.

ANALYSES OF SPIEGELEISEN AND FERROMANGANESE.[3]

Mn	8.11	19.74	41.82	53.32	71.32	80.04	87.92
Si	0.110	0.52	0.42	0.46	1.12	0.97	0.53
P	0.080	0.078	0.100	0.110	0.162	0.175	0.155
C	4.27	4.78	5.63	6.25	6.17	6.53	6.31
Cu	. . .	. . .	. . .	. . .	0.33	0.13	0.23
Fe	87.40	74.75	57.90	39.80	20.65	2.10	4.75
	99.970	99.868	99.870	99.940	99.752	99.945	99.895

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[1] "Journal American Foundrymen's Association," VI, pp. 295–313.

[2] "Transactions American Institute of Mining Engineers," February, 1904.

[3] Turner, "Iron," p. 215.

ANALYSES OF SILICONSPIEGEL.

Si	T. C.	Gr. C.	Mn	P	S	Fe
10.74	2.18	0.33	19.64	0.074	. . .	67.56
12.60	1.65	0.67	19.74	0.080	. . .	66.10
14.19	1.42	1.13	22.98	0.095	. . .	61.60
15.94	1.20	0.90	24.36	0.085	. . .	58.30
12.10	1.70	1.21	18.90	0.105	0.01	. . .

REFERENCES. — Turner, "Metallurgy of Iron," p. 215; "Transactions American Institute of Mining Engineers," XXVIII, p. 783.

In smelting for spiegeleisen[1] it is desirable to have readily reducible ores, high temperature, and a basic slag low in FeO but rich in MnO. In coke blast furnaces as much as 20–30 per cent. of the Mn of the ore goes into the slag. Slags in coke furnaces run SiO_2, 30 per cent.; Al_2O_3, 10 per cent.; MnO, 5–15 per cent. (increasing with richness of pig in Mn); CaO, MgO, 55–45 per cent. With charcoal furnaces the O acid : O basic = 1.4 : 1, hence slags are richer in Mn than when coke is used. Whiting[2] found that in smelting for 30-per cent. spiegeleisen with coke, 1.7 per cent. of the Mn of the ore was volatilized.

In smelting for ferromanganese the ore used is manganese ore instead of a manganiferous iron ore; there is required a very high temperature, hence a light burden, a very basic slag, and a slow smelting. Whiting (*l. c.*) found that in producing 83 per cent. ferromanganese in a coke blast furnace, 5.4 per cent. of the Mn was volatilized. The cause of this is that the temperature at which ferromanganese forms is higher than that at which Mn is volatilized; in the upper part of the furnace the vapor is oxidized by the CO_2 and the water vapor of the furnace gases, or by the O of the ore, and passes off from the furnace as a brownish fume.

Of the eleven analyses of slags made in smelting for ferromanganese given by Wittmann,[3] the following may be quoted with the percentage of Mn of the ferromanganese produced. In calculating the slags by O acid : O base ratio, or by $(SiO_2 + Al_2O_3) : \Sigma$ RO of Platz (see page 77), the author does not come to any satisfactory results. He shows by a diagram how the percentage of Mn in the slag decreases as the sum of CaO + MgO + BaO increases until about 14 per cent. Mn is reached; an additional amount of earth base does no good; on the contrary, makes the slag sticky and consumes fuel.

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[1] Gautier, "Principes de la Métallurgie du Manganèse;" "Bulletin de l'Industrie Minérale de Ste. Etienne," 1889, III[2], p. 878.

[2] "Transactions American Institute of Mining Engineers," XX, p. 290.

[3] "Stahl und Eisen," 1904, p. 15.

Slag.								Ferromanganese.
Mn	SiO_2	Al_2O_3	FeO	CaO	MgO	BaO	S	Mn
23.33	29.02	7.05	0.48	37.03	1.89	. . .	0.67	80
21.09	30.15	7.97	0.56	34.43	3.00	1.88	0.79	55–60
19.18	30.25	6.78	0.52	39.14	2.34	0.19	0.90	50, 70, 80
14.78	30.25	8.32	0.46	39.25	4.65	. . .	1.32	80
13.96	29.25	8.58	0.86	44.01	3.34	. . .	0.55	30–50
12.17	30.91	8.23	0.59	45.62	2.08	. . .	0.63	80

SECTION 28. **Coke (Bituminous Coal, Anthracite) and Charcoal Iron.** Blast furnaces charged with the mineral fuels, bituminous coal, anthracite, and coke, are usually run at a much higher temperature than when charcoal is used. This causes many impurities to be reduced to the metallic state, which at a lower temperature would be slagged; hence coke iron is liable to be less pure than charcoal iron. Further, charcoal furnaces are run with acid slags, which precludes the elimination of S by CaO as CaS; hence the ores used for charcoal furnaces may contain only very little S. Altogether, charcoal furnaces work purer ores than coke furnaces and are run at a low temperature; they therefore produce an iron that is purer and freer from oxide.[1] It was believed that if polished surfaces of coke and charcoal irons were examined with the microscope there would be seen a difference in structure. Of the four photomicrographs[2] shown in Figures 132–135, the coke irons (Figures 134–135[3]) show large particles of ferrite surrounded by long scales of graphite; while in the charcoal iron (Figures 132–133) of a similar composition[4] the particles of ferrite and graphite are smaller and more intimately mixed, making the iron very much stronger. Heyn[5] attributes the different distribution of graphite in the irons to the different rate of cooling. Coke iron, being hotter than charcoal iron, will cool down more slowly than charcoal iron, and thus give the C a longer time to separate out as graphite, and thereby form larger crystals. Sauveur,[6] supporting Heyn's view, gives two photomicrographs of coke iron with Si, 1.54 per cent., and S, 0.075 per cent., cast in sand and chilled, which show differences similar to those of Glinz.

Analyses of charcoal irons, arranged according to their Si content, are given in the subjoined table[7]: —

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[1] Moldenke, "Transactions American Institute of Mining Engineers," XXXI, p. 989.

[2] Glinz, "Stahl und Eisen," 1899, p. 1061.

[3] The composition of Figure 135 being T. C., 3.89; Mn, 0.6; Si, 2.42; S, 0.04; P, 1.0; Cu, 0.04.

[4] T. C., 3.42; Gr. C., 3.17; Mn, 0.77; Si, 2.76; S, 0.02; P, 0.935; Cu, 0.01; Ni, 0.06.

[5] "Stahl und Eisen," 1900, pp. 36, 100.

[6] "Metallographist," III, p. 154.

[7] "Transactions American Institute of Mining Engineers," XXVIII, p. 785.

CHARCOAL IRONS FROM MICHIGAN AND WISCONSIN.

P	Mn	S	Si	Gr. C.	C. C.	T. C.	Per cent. Gr. C.
0.21	0.87	0.010	1.07	3.10	0.71	3.81	81.36
0.19	0.44	0.010	1.20	3.07	0.62	3.69	81.00
0.20	0.76	0.009	1.35	3.20	0.62	3.82	83.76
0.26	0.61	0.010	1.42	2.56	0.52	3.08	83.71
0.24	1.66	0.008	1.56	3.05	0.55	3.60	84.72
0.24	1.07	0.008	1.73	3.35	0.95	4.30	77.90
0.095	0.80	0.003	2.24	3.75	0.30	4.05	92.59

GRADING OF CHARCOAL-IRONS IN MICHIGAN AND WISCONSIN.[1]

PIONEER.			ANTRIM.			ELK RAPIDS.	
Grade.	Si	Chill.	Grade.	Si	Chill.	Grade.	Si
A Scotch.	2.60–2.90	. . .	No. 1 A.	2.00–2.30	. . .	Special 1 A.	2.00–2.20
B Scotch.	2.40–2.60	. . .	No. 1 B.	1.75–2.00	. . .	Special 1 B.	1.80–2.00
C Scotch.	2.10–2.40	. . .	No. 1 C.	1.50–1.75	. . .	Special 1 C.	1.60–1.80
No. 1 soft.	1.90–2.10	. . .	No. 2 D.	1.25–1.50	. . .	No. 1 A.	1.30–1.40
No. 1 special.	1.60–1.90	. . .	No. 2 soft.	1.10–1.25	. . .	No. 1 B.	1.40–1.50
No. 1 foundry.	1.35–1.60	. . .	No. 2 medium.	0.90–1.10	. . .	No. 1 C.	1.50–1.60
No. 2 low.	1.15–1.35	. . .	No. 2 hard.	0.75–0.90	. . .	No. 2 A.	0.70–0.90
No. 2 high.	0.90–1.15	. . .	No. 3 soft.	0.60–0.75	⅛ inch.	No. 2 B.	0.90–1.05
No. 3 low.	0.75–0.95	. . .	No. 3 medium.	0.50–0.60	⅜ inch.	No. 2 C.	1.05–1.30
No. 3 medium.	0.65–0.75	¼ inch.	No. 3 hard.	0.40–0.50	¾ inch.	No. 3 A.	0.50–0.60
No. 3 high.	0.45–0.65	⅝ inch.	No. 4 soft.	0.30–0.40	1 inch.	No. 3 B.	0.60–0.70
No. 4 low.	0.35–0.45	1 inch.	No. 4 hard.	0.20–0.30	1½ inches.	No. 4 A.	0.30–0.40
No. 4 high.	0.25–0.35	1½ inches.	No. 5.	0.17–0.20	Mottled.	No. 4 B.	0.40–0.50
No. 5.	0.10–0.25	Mottled.	No. 6.	0.14–0.17	White.	No. 5.	Under .30
No. 6.	0.00–0.10	White.	. . . .	. . .	. . .	No. 6.	White iron
P		0.15–0.22	P		0.15–0.22	P	0.15–0.22
Mn		0.90–0.60	Mn		0.30–0.70	Mn	0.30–0.70
S		trace–0.018	S		trace–0.018	S	trace–0.018

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[1] "Iron Trade Review," April 14, 1898, p. 7.

On account of the lower smelting power of furnaces using anthracite as compared with coke furnaces, practically all furnaces in the anthracite region have substituted coke for part of the anthracite, usually one-third to one-fourth. Anthracite irons are graded as follows: —

APPROXIMATE ANALYSES OF WARWICK PIG IRON, 1900.

E. B. COOK.

Grade by fracture.	Si	S	P	Mn	Gr. C.	C. C.	T. C.	Remarks.
No. 1 x.	2.00–3.0	0.01–.03	0.4–0.5	0.4–.6	3.4–3.7	0.1–.3	3.5–4.0	Selected carefully by fracture. None shipped by fracture below 2% Si unless especially ordered. 2.0–2.5% Si dark; 2.5–3.0% lighter.
No. 2 x soft.	2.00–3.0	0.02–.04	0.4–0.5	0.4–.6	3.3–3.6	0.2–.4	3.5–4.0	Selected from No. 1 castings. Fracture less open and uniform than No. 1 x. Chemically, practically the same as No. 1 x, except that Gr. C. in No. 1 may average higher. With 2.0–2.5% Si, fracture dark; 2.5–3.0%, lighter and finer crystallization.
No. 2 x strong.	1.50–2.0	0.02–.05	0.4–0.5	0.4–.6	3.3–3.6	0.2–.4	3.5–4.0	Fracture dark and frequently as open as No. 1 x, but grade possibly not so uniformly selected as is No. 1 x; adapted for machinery and structural casting.
No. 2 plain.	2.00–3.0	0.02–.06	0.4–0.5	0.4–.6	3.3–3.5	0.2–.5	3.5–4.0	Selected largely from gray castings; fracture affected by slow running of iron from furnace and other mechanical causes.
Malleable and No. 2 mill iron.	1.00–1.5	0.03–.05	0.3–0.5	0.4–.6	. . .	. . .	3.5–4.0	Iron dark and strong, selected for malleable and car-wheel works; or best bar and sheet iron, horseshoes, and rivet iron.
No. 3.	1.00–1.75	0.04–.08	0.4–0.5	0.4–.6	. . .	. . .	3.5–4.0	The higher Si of this grade selected for pipe foundries or other foundry work requiring close iron; the lower Si for mills.
Mottled.	0.50–0.75	0.08–.125	0.4–0.5	0.4–.6	. . .	. . .	3.5–4.0	No crystallization; fracture gray in color and streaked with white.
White.	0.25–0.75	0.10–.2	0.4–0.5	0.4–.6	. . .	. . .	3.5–4.0	White in color, too hard to drill, brittle in pig.

In Scotland furnaces are run with raw semi-bituminous coal.[1] The furnace gases carry with them tar and ammonia, resulting from the destructive distillation of the coal in the furnace. They are removed by suitable condensing apparatus.

SECTION 29. **Irons Classified According to the Use They Are Put to.** (1) *Acid Bessemer Iron.* This is a gray iron with less than 0.1 per cent. P (Bessemer limit), 1–3.5 per cent. (usually under 2 per cent.) Si, up to 5 per cent. Mn, and low in S, say 0.01–0.09 per cent.

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[1] See Sexton, "Iron Age," November 11, 1897; "Engineering Magazine," March, 1898; Turner, "Iron," pp. 154, 159; Sexton, "Metallurgy of Iron and Steel," 1902, p. 156.

Locality.	Si	T. C.	Gr. C.	Mn	P	S	Remarks.
Hanover[1]	3.31	4.76	4.00	3.41	0.07	0.05	
Duquesne[2]	1.19	. . .	. . .	0.37	0.081	0.018	
Chicago[3]	2.27	. . .	. . .	. . .	0.068	0.014	
Bethlehem[4]	2.00	3.50	. . .	. . .	0.06	0.13	
Steelton[5]	0.78	3.50	. . .	0.58	0.082	. . .	
Sparrow's Point, Md.[6] .	1.30	. . .	. . .	. . .	0.068	0.029	First cast, day
Sparrow's Point, Md.[6] .	0.75	. . .	. . .	. . .	0.069	0.085	Second cast, day
Sparrow's Point, Md.[6] .	1.25	. . .	. . .	. . .	0.063	0.077	Third cast, day
Sparrow's Point, Md.[6] .	1.15	. . .	. . .	. . .	0.066	0.045	First cast, night
Sparrow's Point, Md.[6] .	1.55	. . .	. . .	. . .	0.077	0.064	Second cast, night
Sparrow's Point, Md.[6] .	1.25	. . .	. . .	Average 0.55	0.065	0.032	Third cast, night

[1] Ledebur, "Manual," 4th ed., p. 402.

[2] Private Notes, 1899.

[3] Private Notes, 1893.

[4] Richards, "Notes on Iron," p. 54.

[5] "Transactions American Institute of Mining Engineers," XX, p. 431.

[6] Private Notes, 1901.

(2) *Basic Bessemer Iron.* This is a white iron with 1.5–3 (usually 2–2½) per cent. P, 0.5–2 per cent. Si, 1–2 per cent. Mn, 3 per cent. C, not over 0.12 per cent. S.

	Peine.[1]	Hörde.[1]	Wittkowitz.[1]	Middlesbrough.[1]	Pottstown, Pennsylvania.[2]		
P	1.50–3.0	1.5–2.5	2.00	1.50–2.75	2.43	2.28	2.81
Mn	2.00–2.5	2.2–3.0	1.38	0.70–1.00	1.09	2.00	0.98
Si	0.50–0.7	0.4	0 62	1.70–1.00	0.510	0.394	0.224
S	0.08–0.45	0.1	0.08	0.05–0.12	0.058	0.054	0.025

[1] Wedding, "Basischer Bessemer Process," 1893–1894.

[2] "Iron Age," January 18, 1894.

(3) *Acid Open-hearth Iron.* This is similar to acid Bessemer iron; it only runs lower in Si.

(4) *Basic Open-hearth Iron.* This is similar to basic Bessemer iron; it only runs lower in P, but too high in P to be suited for the acid open-hearth process.

(5) *Foundry Irons.* See above.

(6) *Mill Irons.* This is a broad term for irons not suited for foundry purposes.

(7) *Special Irons.* See above.

SECTION 30. **The Slag.**[1] Many attempts have been made to utilize the waste slag, instead of having to provide room for the ever-increasing size of the dump. Slag is used as road material, as ballast for railroads, and for paving bricks or blocks, when it is cast into molds and then carefully annealed.

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[1] Wood, "Journal Iron and Steel Institute," 1873, p. 186; 1877, II, p. 443; Elbers, "Engineering and Mining Journal," June 3, 17, 1899.

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Slag wool[1] has been made from it by remelting it in a cupola, and blowing into the continuously-issuing small stream of slag a jet of steam. The long, thin threads are drawn off by a fan and deposited in a space covered with wire gauze. The wool is used as a non-conducting, non-inflammable cover for steam pipes, for deafening walls and partitions in houses. Slag wool is not made so extensively as formerly, as in time it is decomposed, hardens into lumps, and the sulphur set free corrodes the pipes. For improved method by Elbers, see "Engineering and Mining Journal," August 26, 1899; "American Manufacturer," December 14, 1899.

The manufacture of Slag Cement[2] has become an important industry. The slag is granulated, dried, ground, and mixed with slaked CaO. According to Detienne,[3] slags suited for this purpose have the following composition: SiO_2, 27–32 per cent.; Al_2O_3, 12–22 per cent.; CaO, 55–49 per cent.; MgO, 1–1½ per cent. A moderate amount of S is harmless.

The subjoined table gives a few analyses of Portland and Slag cements.

	Portland cement, average.[1]	Slag cement, average.[1]	Slag cement from Donjeux.[2]	Slag cement from Saulines.[2]	Slag cement from United States.[3]		
Sand	...	...	0.25	...	...	...	...
SiO_2	20.0–26.5	20.0–25.0	24.55	22.45	28.81	19.46	28.08
Al_2O_3	6.0–14.0	9.0–15.0	14.05	13.95	10 74	10.94	10.80
Fe_2O_3			1.85	3.30			
CaO	58.0–65.5	54.0–60.0	49.25	51.10	55.85	63.50	53.30
MgO	1.0– 3.0	0.6– 5.0	1.60	1.35	1.28	2.08	3.44
SO_3	0.3– 2.4	0.8– 2.6	0.60	0.35	Trace	0.35	1.36
S	...	...	...	...	0.35	...	0.77
Ignition loss	...	...	7.75	7.50	2.31	1.22	2.63

[1] Jantzen, "Stahl und Eisen," 1903, p. 362.
[2] Prost, "Annales des Mines," 1889, XVI, p. 193.
[3] "Engineering Record," 1901, p. 196; "Thonindustrie Zeitung," 1901, p. 617.

Slag cement is manufactured at Chicago, Illinois; Youngstown, Ohio; Sparrow's Point, Maryland.

SECTION 31. **Furnace Gases and Flue Dust.** Analyses of furnace gases given by Ledebur[4] show the following range of composition by volume: N, 55.0–61.5; CO, 21.9–30.1; CO_2, 8.1–22.4; H, 0.01–0.30; CH_4, 0.2–1.9. The averages usually assumed are N, 60; CO, 24; CO_2, 12; H_2; CH_4, 2. Furnace gases contain in addition 5–15 volumes of water vapor, and hold more or less flue dust in suspension. Until recently the furnace gases were burned in the heating stoves and under the boilers. Within the last six years they have been

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[1] "Mineral Industry," 1902, p. 473.

[2] General Articles: Prost, "Annales des Mines," 1889, XVI, p. 158; Detienne, "Revue Universelle des Mines," 1897, XXXIX, p. 237; May, "Stahl und Eisen," 1898, p. 705; Schwarz, "Journal Iron and Steel Institute," 1903, I., p. 203; Detienne, "Annales des Mines de Belgique," 1903, VIII, pp. 33, 406; Jantzen, "Stahl und Eisen," 1903, p. 365; Passow, "Stahl und Eisen," 1903, p. 878; Steger, "Zeitschrift für Berg-, Hütten-, und Salinen-Wesen in Preussen," 1903, pp. 4, 65; Sutcliffe, "American Manufacturer," May 5, 12, 1904. Whiting Process: "Iron Age," March 9, 1899, Curtin Process: "Iron Age," July 17, 1902. Forell Process: "Stahl und Eisen," 1899, p. 1087; 1900, p. 1170. Difference between Portland and Slag Cement: "Thonindustrie Zeitung," 1903, p. 1860.

[3] "Revue Universelle des Mines," 1897, XXXIX, p. 252.

[4] "Manual," 4th ed., p. 643.

used in gas or explosive engines (see page 52). In a gas engine as much as 30 per cent. of the heat energy of the gas is converted into mechanical energy, while with a plain slide-valve engine the utilization is only 5 per cent., with a Corliss engine 6 per cent., a compound condensing engine 8 per cent., and a reheating triple expansion engine 12 per cent., and with superheated steam 16.6 per cent. The main difficulty of the ports in the engines becoming clogged with dust having been overcome (*e.g.*, Stees-Roberts washer), there is a good prospect of using the excess gas of the blast furnace to furnish power for manufacturing purposes. Wehrum is quoted by Grammer[1] as saying, in regard to the Körting gas engines in operation at the works of the Lackawanna Steel Company, Buffalo, New York: "Blast-furnace gas in gas motors is 37 per cent. higher in efficiency than that produced by steam, and I estimate that the engines introduced at the plant of the Lackawanna Steel Company under my administration show an economy of fully 300 per cent. more than that of the single-condensing, steam-blowing engine, which is equivalent to a saving of $12.50 per horse power per year, by the introduction of blast-furnace gas motors. Gas engines are now installed and in operation at the Buffalo plant of the company to the extent of some 5,100 horse power."

The flue dust[2] settled out from the gases was, until recently, simply charged back into the furnace. Since the fine Mesabi ores, forming much flue dust, have formed a large part of the ore charge (from 25 per cent. by the Jones and Laughlin Eliza furnace to 100 per cent.), briquetting machinery (Henry S. Mould Machine, Pittsburgh, Pennsylvania) has been introduced at many plants, which, mixing the dust with a bond (usually slaked lime), compresses it into small cylinders, which, when dried, are charged into the furnace.

SECTION 32. **Cadmies, Lead, Sows, etc.** Cadmia[3] is the name for a zinckose accretion forming at or near the throat of a blast furnace treating zinc-bearing iron ores. The iron ores[4] of western Virginia contain about 0.1 per cent. Zn. They form hard accretions (Zn, 73.00; SiO_2, 2.50; FeO and Al_2O_3, 3.00) at the throat and dusty oxides in the flues of the stoves, which have to be removed at intervals.

Iron ores containing Zn frequently show small amounts of *Lead*, *e.g.*, some ores from Silesia (southeastern Prussia) and Cuba. In Silesia small channels are left open in the brick courses forming the bottom of the crucible, through which the lead trickles out. In Steelton, Pennsylvania, where Cuban ores are smelted, the lead is run out with the pig and collected as far as possible.

Sows, found in the bottom of the crucible when a furnace has been blown down, are, as a rule, lumps of wrought iron, mixed up with more or less slag, showing frequently well-developed crystalline faces similar to spiegeleisen. Occasionally, if titaniferous ores have been smelted, bright red cubic crystalline aggregates[5] of cyanonitride of titanium (Ti_5CN_4) are found with the iron and the slag.

SECTION 33. **Cost of Smelting.** Carroll D. Wright gives, in the Sixth Annual Report of the Commissioner of Labor, "Cost of Production of Iron, Steel, Coal, etc.," for 1890, valuable data. A few examples have been taken from pages 35, 38, 53–55: —

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[1] "Transactions American Institute of Mining Engineers," February, 1904.

[2] Analyses: Lürmann, "Stahl und Eisen," 1898, p. 260; Osann, "Stahl und Eisen," 1902, p. 153; Grammer, "Transactions American Institute of Mining Engineers," XXXIV.

[3] "Iron Age," March 24, 1904; "American Manufacturer," April 28, 1904.

[4] Taylor, "Engineering and Mining Journal," 1899, LXVII, p. 469.

[5] Percy, "Iron and Steel," pp. 163, 510.

Establishment No.	Locality in United States.	Kind of pig iron.	Pig iron per day.	Height of furnace.	Bosh of furnace.	Materials.					Labor.	Officials and clerks.	Supplies and repairs.	Taxes.	Total of all cost.
						Ore.	Limestone.	Coke.	Coal.	Total.					
			Tons.	Feet.	Inches.										
2	North	Hot blast charcoal	73	36	120	$4.007	$0.173	. . .	$ 9.043 a	$13.223	$1.994	$0.261	$0.480	$0.044	$15.202
8	South	Hot blast charcoal	41	55	138	1.981	0.235	. . .	5.541 a	7.757	1.461	0.528	0.498	0.035	10.279
6	North	Cold blast charcoal	10	36	123	8.538	0.330	. . .	11.589 a	20.457	2.932	0.880	0.920	0.060	25.249
20	North	Bessemer	120	70	204	9.009	0.491	$1.750	1.725 b	12.975	1.250	0.091	0.500	0.028	14.844
21	North	Bessemer	236	75	240	10.337	0.430	1.891	. . .	12.658	1.155	0.151	0.513	0.041	14.518
42	North	Foundry No. 1.	81	70	180	3.680	0.517	4.987	1.529 b	10.723	1.983	0.238	0.624	0.023	13.591
55	North	Gray forge	123	70	186	7.229 c	0.676	2.702	. . .	10.607	1.452	0.149	0.587	0.041	12.836

a = charcoal. b = anthracite. c = includes cinder, scrap, etc.

Although the table is old, it shows the relations to one another of the different items of cost. Head[1] gives the cost of smelting 1 ton Bessemer pig at Pittsburgh, Pennsylvania, and Middlesbrough, England, as follows: —

PITTSBURGH, PENNSYLVANIA.		
1.66 tons ore	@ $3.086 =	$5.123
16 cwts. coke	@ .084 =	1.355
12 cwts. limestone	@ .036 =	.433
Labor		.486
Repairs		.243
Other items		.243
Total		$7.88

MIDDLESBROUGH, ENGLAND.		
1.95 tons ore	@ $3.685 =	$7.187
20.5 cwts. coke	@ .188 =	3.855
9 cwts. limestone	@ .459 =	.413
Labor		.729
Repairs		.243
Other items		.243
Total		$12.67

In 1899 the cost of smelting in Pittsburgh, Pennsylvania, was $9–$11 per ton; in Cleveland, Ohio, $1 less. Kirchhoff[2] gives $10.82 per ton of Bessemer pig as an average for 1901. E. Phillips[3] gives the cost of smelting in England with Bilbao (Spain) 55 per cent. ores, and in the United States with Lake Superior 65 per cent. ores, as $12.50 and $7.50 per ton of Bessemer pig. W. B. Phillips[4] gives several detailed accounts of the cost of producing pig iron in the United States. Odelstjerna[5] gives the cost of making 1 ton charcoal iron at the Hinkle Works, Ashland, Wisconsin, in 1896, as follows: ore, $4.37; charcoal, $5.29; limestone, $0.09; labor, $1.17; repairs, other items, $0.85; total, $11.77.

Birkinbine, in a paper on comparative costs,[6] reviews the decrease of cost in producing pig iron within the last twenty years; so does Kirchhoff, in a presidential address before the American Institute of Mining Engineers.[7] The cost of building a new iron blast-furnace plant in the Central Western States may be estimated to be: tons product in twenty-four hours multiplied by $1,500–$2,000.

SECTION 34. **Refining of Cast Iron.** When cast iron contains impurities such as Si, Mn, P, and S, which make it unsuited for certain purposes, they can be removed by a special refining process, without oxidizing the C. Most of these processes have become obsolete.

(1) *Silicon*, with it *Manganese*, was removed by melting down the pig in a reverberatory furnace and blowing air on to it, or by melting it down with coke in a hearth furnace having tuyères pitching downward[8] and a basic cinder rich in iron.[9] Since furnace men have learned to control the Mn and Si of the pig in blast-furnace work, the refining has lost its former importance (see, however, Section 43, following).

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[1] "Cassier's Magazine," October, 1899; XVI, p. 645.

[2] "Iron Age," November 14, 1901, p. 13.

[3] "Engineering Magazine," 1901, XXI, p. 185.

[4] "Engineering and Mining Journal," August 31, 1901.

[5] "Stahl und Eisen," 1896, p. 351.

[6] "Iron Age," March 29, 1896.

[7] "Transactions American Institute of Mining Engineers," 1899, XXIX, p 352.

[8] "Run-out Fire or Refinery."

[9] See Percy, "Metallurgy of Iron and Steel," p. 621.

(2) *Phosphorus* can be removed from pig iron by melting it down in a reverberatory furnace lined with iron ore, in the presence of a basic ferruginous slag and oxidized iron, and keeping it at a temperature which does not exceed much its melting-point. If the temperature is too high, the irony slag is decomposed, the hearth strongly attacked, and the P forced back into the metal. With the P, the Si and Mn and most of the S are also removed. This process, invented by Bell[1] and independently by Krupp, goes by the name of *Krupp Pig-washing*. At Essen, Germany, it was carried out in a Pernot reverberatory furnace[2] heated with gas and provided with regenerative chambers. The characteristic of the Pernot reverberatory furnace is a circular (12 feet diameter, 3 feet deep) hearth revolving in an oblique plane. In this country the process was used at the works of the Cambria Iron Company as late as 1893, at the Youngstown Smelting and Mining Company, and perhaps other works, but has been almost entirely replaced by the basic open-hearth process. At the works of the Pennsylvania Steel Company, Steelton, Pennsylvania,[3] the process is carried on in a modified way. Pig iron, liquid or solid, is charged into a basic open-hearth furnace and freed almost wholly from Si, Mn, P, and from part of the S. It has now assumed the character of washed metal, is tapped into a ladle and poured into an acid open-hearth furnace. If washed metal, *i. e.*, white iron low in Si, Mn, P, and S, containing 2 + per cent. C, is to be sold to crucible steel or acid open-hearth plants, the refined metal from the basic open-hearth would be cast into chills. Micrographs of washed metal have been published by Howe[4] and Campbell[5].

(3) *Sulphur* is readily removed from iron by means of Mn, which has a greater affinity for it than Fe has. The affinity of Mn for S was first utilized in 1890, at Hoerde, Germany, where low-silicon iron with not less than 1.5 per cent. Mn is tapped from the blast furnace into a ladle and poured into one of a pair of mixers[6] (Figure 136) having a capacity of 100 to 180 tons, where it remains at least twenty minutes. The MnS formed rises to the surface and is taken up by the slag. The following tables, taken from "Ledebur's Manual," 4th ed., p. 699, show the effectiveness of the process and the composition of the resulting slag.

Average of		Taken in	Si	P	Mn	S	
15	Blast-furnace samples	1 day	0.24	2.22	2.70	0.137	Hoerde
	Mixer samples	1 day	0.23	2.13	2.42	0.038	
13	Blast-furnace samples	1 day	0.31	2.16	2.16(?)	0.111	Hoerde
	Mixer samples	1 day	0.27	2.02	2.16	0.040	
23	Blast-furnace samples	1 month	Not determined	Not determined	Not determined	0.163	Hayingen
	Mixer samples	1 month	0.22	2.80	1.46	0.060	

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[1] "Journal Iron and Steel Institute," 1878, p. 17.

[2] Holley, "Transactions American Institute of Mining Engineers," VIII, p. 156.

[3] Campbell, "The Manufacture and Properties of Iron and Steel," 2d ed., p. 302.

[4] "Iron, Steel and Other Alloys," p. 188.

[5] "Proceedings American Society for Testing Materials," 1903, III, p. 176.

[6] "Berg- und Hüttenmännische Zeitung," 1892, Plate I, Figure 4.

Slag from	SiO_2	Al_2O_3	MnO	MnS	FeO	CaS	CaO	MgO	P_2O_5
Hoerde . .	18.90	5.00	20.23	28.01	25.46	3.53	. . .	0.43	Not determined
Hayingen . .	35.70	Not determined	36.38	14.11	5.01	. . .	Not determined	Not determined	Not determined
Oberhausen .	Not determined	2.46	45.22	19.02	6.78	. . .	2.58	0.19	0.31

SECTION 35. **Electric Reduction of Iron Ores.**[1] The reduction of iron ores by electricity to form pig iron can be considered only in connection with the occurrence of a rich, pure ore in proximity to cheap water power and the production of a fancy grade of pig iron to be converted into a special brand of steel. Ordinary pig iron cannot be produced under any normal economic conditions. According to Keller[2] one kilowatt year (K) utilized in an electric reducing furnace will yield 4 tons pig iron. Assuming 350 kilograms coke to be necessary for the reduction of the iron ore, the electric energy absorbed and the reducing coke consumed per ton of pig will be: $\frac{K}{4} + 350$. In the blast furnace 1 ton coke is required to produce 1 ton of pig iron. Assuming that the labor, fluxes, and general expenses be the same in both processes, that the cost of producing blast be equal to that of supplying electrodes, and that the cost of coke at the blast furnace is \$1.50 a ton and at the power station \$3, then $\frac{K}{4} + \frac{\$3 \times 350}{1{,}000} = \$1.50$, or $K = \$1.80$, *i.e.*, one kilowatt year may cost only \$1.80 if electric smelting is to compete with blast-furnace smelting. At Niagara one horse power year costs \$18, at the Merrimac River \$30, in Boston (coal @ \$3 per ton) \$80; therefore one kilowatt year = 1.35-horse power year costs \$24.30 at Niagara, \$29.70 on the Merrimac River, and \$108 with steam power in Boston.[3] The smelting of iron ore for pig iron was first carried out by Stassano in Italy,[4] but he soon changed his process to the conversion of the fluid pig iron into steel. Such combination methods have been proposed and are in part being worked in Italy and southeastern France by Stassano,[5] Héroult,[6] Harmet,[7] Keller,[8] and others.

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[1] Borchers, "Electrometallurgie," 3d ed., p. 524; "Berg- und Hüttenmännische Zeitung," 1903, pp. 481, 494; "Journal Iron and Steel Institute," 1903, I, p. 170; Garnier, "Electrosidérurgie," Paris, 1903.

[2] "Journal Iron and Steel Institute," 1903, I, p. 170.

[3] J. W. Richards ("Technology Quarterly," 1904, XVII, p. 30) says that "electric heating will cost less where coal costs per ton over one-half of the price paid per electric horse power year."

[4] "Berg- und Hüttenmännische Zeitung," 1903, p. 482.

[5] "Electro-chemical Industry," I, pp. 247, 363.

[6] *Op. cit.*, I, pp. 63, 287, 449.

[7] "Comptes Rendus, Industrie Minérale, Ste. Etienne," 1902, p. 85.

[8] "Journal Iron and Steel Institute," 1903, I, p. 176; "Electro-chemical Industry," I, p. 162; "Iron Age," June 9, 1904.

CHAPTER III.

FOUNDRY PRACTICE.

SECTION 36. **Literature**: Manuals quoted on page 1, then:—

Bolland, S., "The Iron Founder and Supplement." New York, 1893.
Dürre, E. F., "Handbuch der Eisengiesserei." Leipsic, 1890.
Guettier, A.; "Fondeur en Métaux." Paris, 1890.
Ledebur, A., "Handbuch der Eisen und Stahlgiesserei." Third edition, Weimar, 1901.
West, Th. D., "American Foundry Practice." New York, 1900.
West, Th. D., "Moulder's Test Book." New York, 1899.
Kirk, E., "The Cupola Furnace." Philadelphia, Baird and Company, 1903.
Periodicals: "Journal of the American Foundrymen's Association." New York.
"The Foundry." Cleveland.

SECTION 37. **Introductory.** In 1902[1] 21.6 per cent. of the country's product of pig iron was used for foundry purposes. In the same year there were 4,615 foundries in the United States,[2] of which nearly 50 per cent. were in Pennsylvania, New York, Ohio, Illinois, and Michigan.[3]

In foundry work, cast iron is melted down in a furnace and the liquefied metal cast into molds. The iron, the furnace, and the mold are variable, according to the casting that is to be produced.

SECTION 38. **Quality of the Iron.**[4] With the exception of special cases, as in making hard castings and malleable castings when white iron is required, all castings are made of gray iron. Foundry irons Numbers 1–4 are the irons commonly used, *i.e.*, Number 1, Si, 2.75 per cent., S, 0.035 per cent.; Number 2, Si, 2.25 per cent., S, 0.045 per cent.; Number 3, Si, 1.75 per cent., S, 0.055 per cent.; Number 4, Si, 1.25 per cent, S, 0.065 per cent. (see pages 91 and 94). As regards the phosphorus content, the founder distinguishes three classes: <0.4 per cent. P (gun iron, car wheels, specification castings); 0.4–0.8 per cent. P (jobbing work, machinery, gray castings in general); >0.8 per cent. P (art work, stoves, novelty castings). With manganese the line of division lies at 0.8 per cent.: <0.8 per cent. Mn for

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[1] "Mineral Industry," XI, p. 361.

[2] "The Foundry," February, 1902, XIX, p. 230.

[3] American foundries: "Stahl und Eisen," 1902, pp. 930, 990; 1903, pp. 240, 248, 260, 325. "Foundry," XIV, p. 97, General Electric Company; XIV, p. 193, Walker and Pratt Manufacturing Company; XVII, p. 141, Westinghouse Manufacturing Company; XX, p. 5, Mesta Machine Company; XXII, p. 214, Sheffield Car Company; XXIII, p. 100, B. F. Sturtevant Company; XXIII, p. 155, Allis-Chalmers Company; XXIV, p. 51, Acme Foundry Company. "Iron Age," May 30, 1901, McCormick Foundries; July 9, 1903, Brown and Sharpe Manufacturing Company; October 29, 1903, B. F. Sturtevant Company; May 19, 1904, Model Foundry, St. Louis Exposition. "Cassier's Magazine," 1902, XXI, p. 292, Modern Foundries. "Engineering Magazine," 1902, XXIV, pp. 369, 515, 695; 1903, XXV, pp. 49, 215, 409, Management.

[4] Scott, "Iron Age," November 10, 1900.

ordinary, >0.8 per cent. Mn for special castings. The influences of the different elements and their right proportions have already been discussed (page 87). Thus, soft iron must have much graphitic carbon; hard iron more combined carbon; strong iron must be pure, run low in graphite, and contain not less than $1\frac{1}{2}$ or over $2\frac{1}{2}$ per cent. Si. Iron that is to resist chemicals must contain much combined carbon or Si. As to the combined carbon, Ledebur[1] gives, in the subjoined table, the percentages of iron dissolved by treatment for sixty-five hours with dilute H_2SO_4. The table shows that cast irons which are strong and tough do not resist chemicals successfully.[2]

Spiegeleisen.	White iron.	Coke iron No. 1.	Gray charcoal iron.	Steel.	Wrought iron.
14.1	19.7	27.6	37.7	66.5	88.6

The shrinkage, *i.e.*, the contraction of cast iron while cooling, is of greatest importance. Sometimes shrinkage is called the diminution in volume which takes place at the moment of set, when the metal is spongy, and contraction the diminution after the moment of set, when the red-hot metal cools down. Gray iron at the moment of solidification expands, on account of the crystallization of graphite, giving a sharp impression of the mold, and then diminishes in volume as it cools down. In practically all cases the contraction is greater than the previous expansion, so that the casting is always smaller than the pattern around which the mold was made. The amount of shrinkage varies greatly with the grade of the iron. The presence of Mn, being favorable to the formation of combined carbon, increases the shrinkage. A dead-white iron contracts a little over $\frac{1}{4}$ inch to the foot. Si, being favorable to graphitic carbon, decreases shrinkage; P decreases it. A dead-gray iron contracts a little under $\frac{1}{8}$ inch to the foot. It is customary to allow with gray castings $\frac{1}{8}$ inch for 1 lineal foot, which makes the shrinkage coefficient $\frac{1}{96}$.

When iron is poured into a cold mold, solidification begins at the outside and progresses toward the centre. The crystals forming will arrange and group themselves[3] with their principal axis in lines perpendicular to the cooling surfaces, as shown in Figures 137–139. Figure 139 shows why the casting is liable to be weak along line *ab* and strong along *cd*. A fillet in the corner would to some extent correct the weakness along *ab*, and machining the casting would furnish the sharp corner. The casting at first consists of a comparatively cool, rigid hull, containing hot, soft material. With a cube the contraction will proceed evenly toward the centre; with a narrow, rectangular piece it will be greater at the ends than at the sides. The uneven shrinkage is counteracted to some extent by the use of a "riser," a gate set on top of the casting or leading upward from the side, which serves to supply the fluid metal necessary to make good the deficiency caused by the shrinkage.[4] The riser further serves to show that the

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[1] "Eisen und Stahlgiesserei," 3d ed., p. 69.

[2] Standard Specifications: "Transactions American Institute of Mining Engineers," February, 1904.

[3] "Journal American Foundrymen's Association," 1900–01, IX 1, p. 33.

[4] "Foundry," XII, pp. 96, 99.

7

mold is full and to free the metal from drosses (dirt). The riser[1] should be placed on the heaviest part of the casting, and should be of sufficient bulk to hold enough metal to feed the casting satisfactorily. The height of the riser is of little importance. In Figure 140 (*ibidem*) the casting, which has been poured through the gate, will begin to cool at *A*; this upon solidifying will draw metal from *B*, *B* from *C*, and so on to *F*. Now the surface of *F* was cooling, while the parts *A* to *E* were solidifying; but the interior will be still liquid when parts *A* to *E* have become solid. If *F* now solidifies without having a riser to feed the necessary metal it can draw only from the liquid interior; the result is a shrink hole *H*, or a depressed surface, or both.[2] Variations in the shrinkage are easily observed in casting a wheel (Figure 141); the outer ring, the spokes, and the hub all will contract differently. If the outer ring is very thick, in comparison with the spokes and the hub, the spokes will solidify first (but will not contract in the direction of their length, as there remains liquid iron at either end). Then the hub will become rigid, and when, last of all, the ring solidifies and contracts, it will exert a great pressure on the spokes, which may crush them; or, if the spokes are thick enough, the ring may be fractured. In any case, an inner tension is set up which weakens the casting. If the ring and spokes are thin and the hub thick, ring and spokes will solidify and shrink, and when the hub contracts, the spokes will snap and break away from it. Thus in casting, the pattern must be so made that the iron shall be evenly distributed; or an iron be chosen that will shrink little; or a casting must be made in pieces and joined together (large fly wheels). In shrinking away from the mold, castings are liable to become porous (lower side of stove plates), but porosity is caused also by the occlusion of gases.[3] These may be air which could not escape from the mold excepting through the metal (improper venting), or H, N, and to a small extent CO, absorbed by the iron. Over 1 per cent. Mn favors the absorption of gases; Si and C are unfavorable to gas absorption. As the absorption increases with the pressure, blast-furnace metal contains the most gas, cupola metal less, and iron melted down in the reverberatory furnace, none at all. For most foundry purposes is needed an iron which (1) shows a gray fracture; (2) forms no finery scum; (3) is easily worked by chisel and file; (4) fills the mold even to the thinnest sections; (5) chills with a smooth surface; (6) has moderate strength; (7) is free from blowholes; (8) shrinks but little. These conditions are well filled by an iron containing, after melting and casting, C, 3.5 per cent.; Si, 1.5 per cent. to 2.0 per cent.; P, not over 0.8 per cent.; Mn, under 1 per cent.; S, a trace. When such a mixture has been obtained variations may be made. For large castings, lower the Si (because the slow cooling gives the C a good chance to separate as Gr. C.); for small castings increase it. If shrinkage is to be avoided, lower the Mn. If much strength is required, lower the Gr. C., Si, Mn, and avoid P. To prevent blowholes reduce S and increase Mn. To prevent kish increase scrap or raise Mn. If resistance to chemical action is required, make Mn as high as brittleness will allow. Open-bright and silver-gray pig irons and ferrosilicon are used to increase the Si; spiegeleisen, ferromanganese, and pig iron rich in Mn are used to increase the Mn; scrap, both wrought iron and steel, is used to dilute carbon, etc.[4] In order to test iron or iron mixtures

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[1] "Metal Industry," 1904, II, p. 19.

[2] Skimming gates, see "Foundry," 1902, XX, p. 18.

[3] Wickhorst, "Foundry," 1899, XIII, p. 280; Münker, "Stahl und Eisen," 1904, p. 23.

[4] Analyses of foundry irons used for various purposes: Meissner, "School of Mines Quarterly," X, p. 147; "Iron Age," April 14, 1895; November 1, 1900; all through "Foundry," and "Journal American Foundrymen's Association."

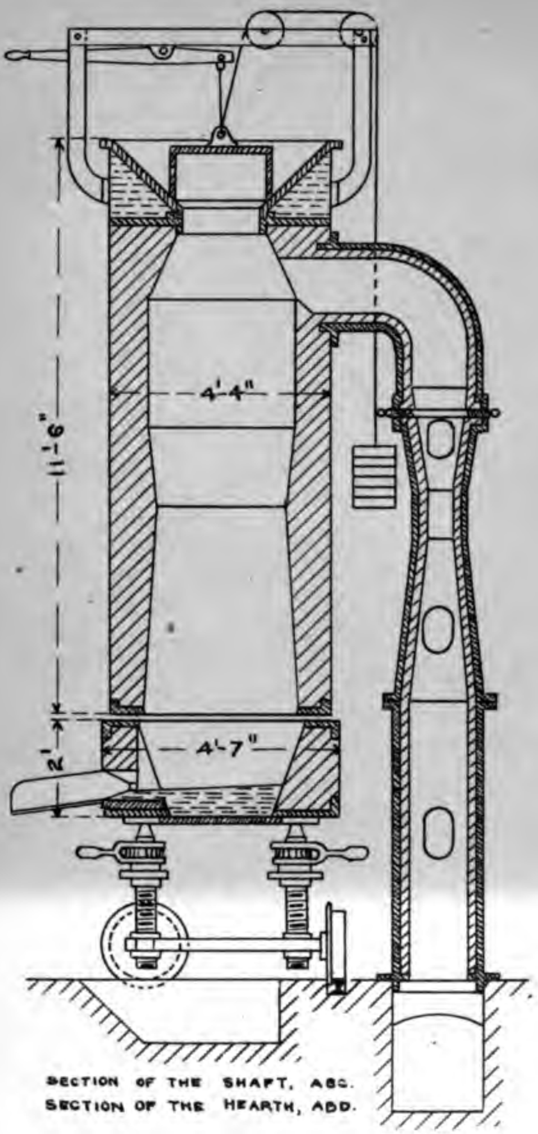
[illegible]

[illegible]

[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]

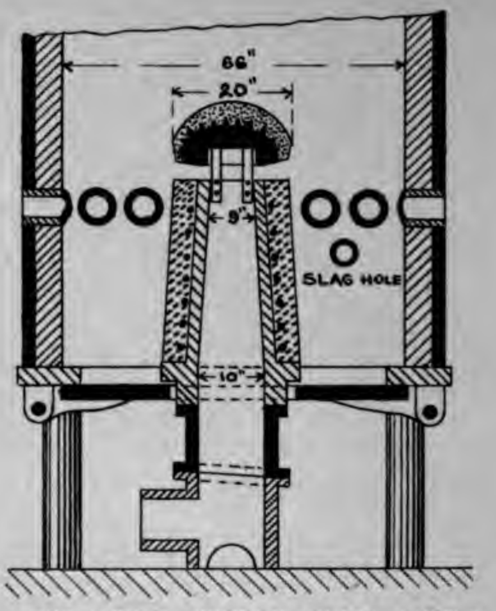
[illegible]

FIG. 148.



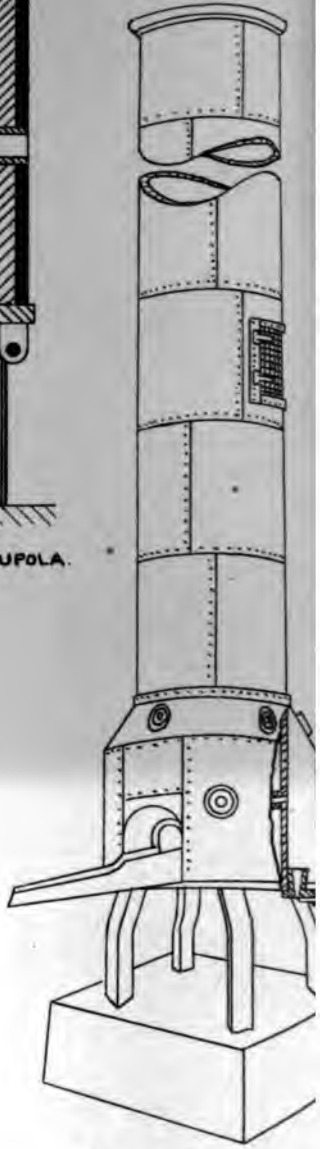
SECTION OF THE SHAFT, ABC.
SECTION OF THE HEARTH, ABD.

FIG. 145.



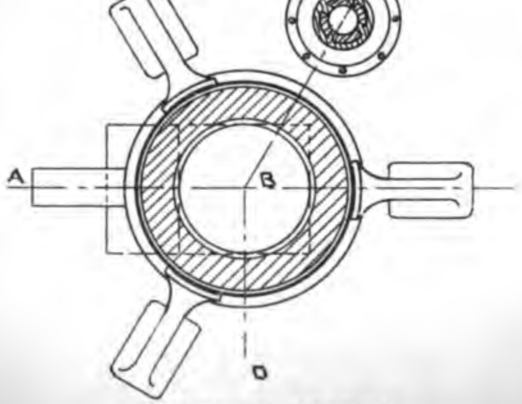
THE WEST CENTER BLAST CUPOLA.

FIG. 143.



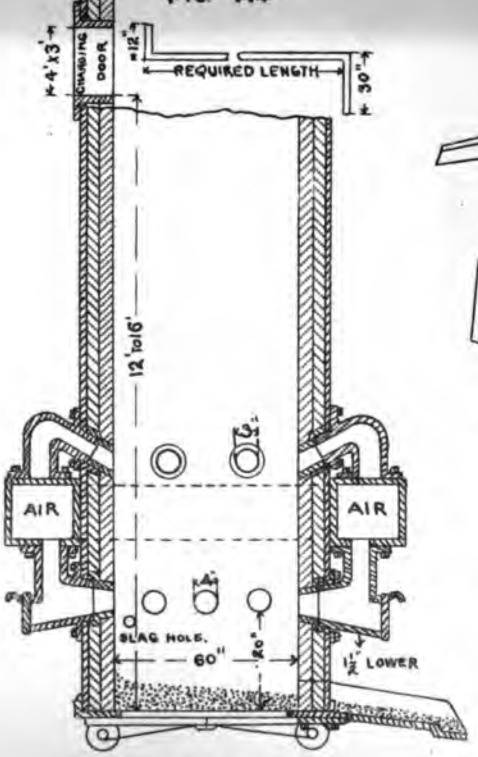
THE PAXTON- COLLI
CUPOLA.

FIG. 147



HORIZONTAL SECTION.
THE HERBERTZ DRAUGHT CUPOLA.

FIG. 144



THE JEWETT CUPOLA.

Inside diameter.	Height.	Charging door.	Melting capacity.	
Inches.	Feet.	Inches.	Tons per hour.	Tons per heat.
18	6–7	15 x 18	¼–¾	1–2
20	7–8	18 x 20	½–1	2–3
24	8–9	20 x 24	1–2	3–5
30	9–12	24 x 24	2–5	4–10
40	12–15	30 x 36	4–8	8–20
50	15–18	30 x 40	6–14	15–40
60	16–20	30 x 45	8–16	25–60

A cupola usually has an internal crucible, and is worked with blast; occasionally the crucible is external,[1] or it is worked with forced draught.[2]

Description of Cupolas. The Paxson Hot-Blast Colliau Cupola[3] is shown in perspective in Figure 143. It is a cupola resting on an iron ring, supported by four legs. It has two rows of tuyères, which receive hot air from an air box surrounding the crucible, a drop bottom, and a screened charging door. The Jewett Cupola[4] is shown in vertical section in Figure 144. The West Centre Blast Cupola, shown in Figure 145,[5] has a tuyère in the bottom in addition to the single row at the sides, in order to obtain a more perfect combustion. The Newton Cupola, see "Iron Trade Review," June 6, 1901, and papers of Kirk and Moldenke quoted. The Herbertz Draught Cupola[6] is shown in Figures 146 and 147. It consists of three parts: the shaft, with closed top supported by three legs; the crucible, with hinged bottom mounted on four screws (turning in nuts fixed in a carriage), which by their common action permit enlarging or reducing the annular slot through which air is drawn in; and the smoke pipe with steam-jet applied at the centre, creating in the upper region of the cupola a vacuum of 3–4 inches water. The shaft has two peep holes through which the fire can be watched. Along the bottom of the crucible are placed three tuyères used in blowing in. The advantages claimed for working by draught are: cheapness of plant (no machinery only a boiler is necessary); low fuel consumption (1 coke : 20 pounds iron on account of perfect combustion of C to CO_2); little oxidation of C and Si; little loss in iron (3½ *versus* 6 per cent.); large capacity (2 tons per hour). At Elizabethport, New Jersey, the total charge of a day was: filling coke 576 pounds, 6 charges of melting coke @ 72 pounds = 432 pounds, 6 limestone @ 15 = 90 pounds, 6 Number 3 foundry iron @ 300 = 1,800 pounds, and 6 foundry scrap @ 900 = 5,400 pounds; total iron 7,200 pounds. The hearth, prepared and dried outside, is filled with shavings, wood, and coke, then pushed under the shaft and raised until it comes in contact with the lower rim of shaft. Coke is charged to the upper peep hole and the fire started through the three tuyères.

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[1] Krigar cupola, "Engineering and Mining Journal," February 16 +, 1889; "Stahl und Eisen," 1903, p. 1077.

[2] Herbertz cupola, see below.

[3] Pier 45, North, Philadelphia.

[4] "Foundry," 1899, XV, p. 61.

[5] "Journal Iron and Steel Institute," 1897, I, p. 276.

[6] "Iron Age," June 18, 1891.

When the filling coke is ignited, the charges given above are fed, the hearth is lowered $1\frac{1}{4}$ inches, the tuyères are plugged with molding sand, the cupola is allowed to work by natural draught ($\frac{1}{16}$ inch vacuum) until the first iron appears; and lastly the steam jet is put on, giving a vacuum of 3–4 inches water. The 7,200 pounds melt in one hour and twenty-four minutes. When the last iron is run from the cupola the bottom door is opened, the residue falls into the ash pit, is quenched with water, and the good coke picked out. In the Herbertz cupola for melting steel[1] the air is sucked in at the top, then down through vertical pipes placed between the iron shell and the fire-brick lining, and passes into the furnace, after having become superheated to 260–600° C. The furnace is used in Europe, but not yet in this country.

Blast Cupolas. The melting power of the blast cupola depends upon the size of the furnace, the pressure, amount,[2] and even distribution of the blast,[3] the kind and quality of the fuel, the distribution of the charge, etc. Jewett[4] gives, as a safe rule to allow for a cupola, 24–40 inches in diameter and a blast pressure of 8–10 ounces per square inch in the bustle pipe: with anthracite 7 pounds iron per hour per square inch of hearth area, and with coke $9\frac{1}{2}$ pounds. The actual performance of many cupolas is above this, especially with an inner diameter greater than 40 inches, and with two rows of tuyères. Most modern cupolas are of the Colliau type, *i. e.*, they have two rows of tuyères 16–20 inches apart, as this secures a more perfect combustion, and thereby better utilization of the fuel. With cupolas up to 40 inches in diameter the first row is 18–20 inches above the bottom, and 26–30 inches with cupolas 40–72 inches in diameter and larger. One of the tuyères of the lower row should be $1\frac{1}{2}$ inches lower than the rest, so as to permit watching the level of the metal, and thus insure tapping at the right moment. With a double row of tuyères the combined tuyère area should be 25 per cent. greater than that of the bustle pipe. The Greiner cupola[5] has a large number of small tuyères arranged in spiral form.

The fuel[6] should contain not over 0.7 per cent. S, as much S, even in the presence of considerable lime, will enter the iron, the amount taken up[7] being inversely proportional to the temperature. The rate at which S enters the iron is shown by the experiments of Jüngst,[8] who found that, in six remeltings with a good coke and sufficient lime, the sulphur of the iron rose from 0.06 to 0.20 per cent. S is in part carried away by the slag by adding manganese to the charge.[9]

The charge,[10] *i. e.*, iron (broken and placed radially), cast-iron scrap, steel scrap (up to 25 per cent.),[11] and coke with limestone[12] (about 40 pounds limestone per ton of iron) giving slags

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[1] "Iron Age," June 18, 1891, p. 1154.

[2] Calculation: Lürmann, "Stahl und Eisen," 1891, p. 390.

[3] Cupola Fan Practice: "Iron Age," June 23, 1904.

[4] "Foundry," 1899, XV, p. 60.

[5] Durfee, "Engineering and Mining Journal," February 16, 1899; Ledebur, "Eisen- und Stahlgiesserei," 3d ed., p. 110.

[6] Coke: "Foundry," 1901, XVIII, p. 261; 1903, XXII, p. 204; "Iron Age," November 1, 1900, p. 10. Anthracite: "Foundry," 1901, XVIII, p. 111.

[7] Howe, "Transactions American Institute of Mining Engineers," XXXI, p. 990.

[8] Ledebur, "Eisen- und Stahlgiesserei," 3d ed., p. 150.

[9] "Stahl und Eisen," 1902, p. 415; 1903, p. 1135; "Iron Age," November 9, 1903, p. 14.

[10] Management of cupola: "Foundry," 1902, XX, p. 51.

[11] Diller, "Journal American Foundrymen's Association," 1902, XI^1, p. 25; Moldenke, "Transactions American Institute of Mining Engineers," February, 1904.

[12] "Stahl und Eisen," 1904, p. 28; "Iron and Steel Metallurgist," I, p. 309.

with 46–50 per cent. SiO_2,[1] should be evenly distributed. In starting a cupola[2] a fire is lighted, a bed of coke (anthracite) is given, and the furnace filled with charge. Some melters fill the cupola and allow the fuel to burn by natural draught until a few minutes before the iron is wanted, and then put on the blast; others put on the blast while filling and increase its volume until the metal begins to flow. The amount of bed coal varies with the diameter of the furnace. A good rule[3] is to have it 18–20 inches above the lower row of tuyères for anthracite, and 20–23 inches for best Connellsville coke. These figures are correct with a blast pressure of 12–14 ounces, slightly too large with 6–8 ounces. The first iron charged should be four to six times the weight of the bed coal; the second fuel charge is about 10 per cent. of the bed charge, and the iron the same as before, and so on, using the same proportion of pig and coke. After a few melts slight changes in these proportions may be called for. The pig iron passing through the cupola is warmed by the ascending gas current, softens, and is slowly liquefied; it trickles in drops through the incandescent coke, and becomes more or less oxidized by the action of CO_2 and O. The denser the fuel and the smaller the amount charged the stronger will be the oxidation and the greater the melting loss of iron. While blowing in, the tap hole is kept open (that the bottom may become thoroughly heated) and iron allowed to run out until it comes freely and hot. Then only is the hole plugged. The first iron is usually not very hot, having been cooled by the sand. It is put aside to be melted over again. The hot metal[4] is run into small- or large-size ladles,[5] suited to different-sized work, and cast. When no iron is to flow, the tap hole is stopped by a bod-stick — a stick 1½–2 inches in diameter and 5–10 feet long, with a bod (a ball of clay, loam, or steep). When a melt (heat) is finished, say at 2 P.M., the blast is stopped, the hinged doors are dropped, the sand and refuse fall on the floor or into the pit, are chilled with water, and any good coke and iron removed;[6] the sand may be sifted out from the slagged parts which go to the dump. Any crusts adhering to the sides of the lining are removed while hot. When the cupola is cool, it is daubed with clay or repaired if necessary. The loss in melting is about 6 per cent.[7]

Remelting in the Reverberatory Furnace is suited especially for large castings. It is always practiced with charcoal iron, as in melting this in the cupola it takes up carbon and loses one of its characteristic advantages of running low in carbon. The reverberatory furnace permits a better control of the grain of the metal than the cupola, and causes fewer blowholes to form; but the cost of plant and of fuel is higher. It is used, therefore, only under special circumstances. The metal, being subjected to stronger oxidizing conditions in the reverberatory furnace than in the cupola, undergoes greater change in the melting down. Thus Mn, Si, and Fe are somewhat oxidized, C only when most of the Mn and Si have been slagged. If C is to remain unchanged, the pig iron must run high in Mn, say 2 per cent.; P remains

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[1] Ledebur, "Eisen- und Stahlgiesserei," 3d ed., p. 151.

[2] "American Machinist," February 12 (Smith), March 5 (Wolcott), 1891; "Iron Age," April 14, 1892 (H. R. H.); June 3, 1897 (Beckett).

[3] Jewett, "Foundry," 1899, XV, p. 60.

[4] Longmuir, "Journal Iron and Steel Institute," 1903, I, p. 457; Moldenke, "Transactions American Institute of Mining Engineers," February, 1904.

[5] "American Machinist," August 4, 1892.

[6] Keep, "Iron Age," June 9, 1898; Sey, "Iron Trade Review," May 2, 1901.

[7] "Journal American Foundrymen's Association," 1901, X^2, p. 147.

unchanged; S, which would be somewhat eliminated by the Mn, is increased in most cases, as the iron takes up S from the fuel gases.

Reverberatory furnaces have deep or shallow hearths. A furnace with a deep hearth is shown in Figure 147.[1] The bed, 14 x 4½ feet, slopes from flue- to fire-bridge, near which is the sump with the tap hole. The bed is usually rammed with sand and clay or ground brick and clay; it lasts two to three heats. On one side of the furnace are a large charging door and three working doors. The metal bath of 12–15 tons is 6–12 inches deep. According to Ledebur,[2] there is required for 1 ton metal a hearth area of 5½–6½ square feet for large furnaces with over 5-ton charges, and 8½–10½ square feet for smaller furnaces; the length is 10–13 feet; the width is the same as that of the fire-bridge; the grate area is one-third the hearth area; the length : the width = 1 : 2 = 1 : 2.75; the depth of grate is about 18 inches; the flue above the fire-bridge is 0.7, the flue leading to stack, $\frac{1}{8}$–$\frac{1}{10}$, and the stack 0.5 the area of the grate. The stack is about 80 feet high. The furnaces are usually fired with a long-flame, bituminous coal and natural or forced draught. Using coal, a heat with a cold furnace lasts about six hours; with a warm furnace, three hours. The coal consumption is 50–75 pounds per 100 pounds iron; the melting loss, 5–8 per cent.

5 | 45·0 | 55

Reverberatory furnaces with shallow hearths, *i. e.*, hearths sloping from fire- to flue-bridge,[3] are not as much used as those having a sump near the fire-bridge, as for a given amount of metal the oxidizing effect of the flame is much greater.

Oil- or gas-fired furnaces with Siemens regenerative chambers are finding favor in foundries having to make very large castings. Stanford,[4] in melting 8-ton charges in a Siemens furnace with three oil burners at either end, found the time required for melting to be two hours, and the oil consumption 450 gallons in twenty-four hours. Of course only the air was superheated in the regenerators.

Before charging a direct-fired furnace, the bottom is covered with inch boards for protection. At first light scrap and pig are charged, which melt readily; then follow the heavy parts of the charge. All iron should be piled openly to permit the hot gases to pass through the open spaces in the charge. When melted, the iron is poled for five to ten minutes to stir it up and make it uniform, and to bring drosses to the surface. The character of the melted iron is ascertained by casting in sand sprues, say 6 inches long, 7 inches diameter at top and $\frac{5}{8}$ at bottom, and noting the fracture.

SECTION 40. **Character of Mold.** Molds are made of green sand, dry sand, loam, and iron (chills).

Green sand contains about 94 per cent. SiO_2, 5 per cent. Al_2O_3, 1 per cent. Fe_2O_3, CaO, MgO, alkalies. The Fe_2O_3 may reach 3–4 per cent.; CaO ought always to be very low. The subjoined analyses deviate somewhat from this standard.

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[1] West, "American Foundry Practice," 1900, p. 339.

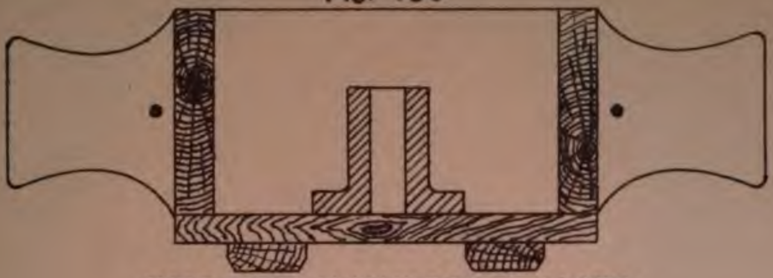
[2] "Eisen- und Stahlgiesserei," 3d ed., p. 100.

[3] "Transactions American Society Civil Engineers," 1895, XXXIV, p. 1; "Iron Age," p. 15, 1895.

[4] "Transactions American Society of Civil Engineers," 1893, XXXIV, p. 1.

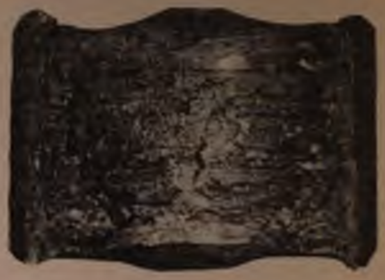
THE N
PUBLIC
ASTOR
TILDEN

FIG. 150



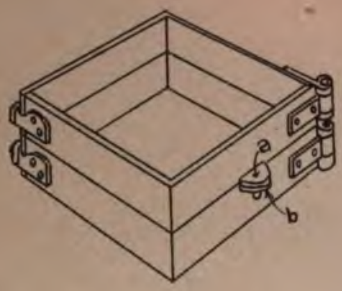
FIRST STEP IN MOULDING A GLAND.

FIG. 162.



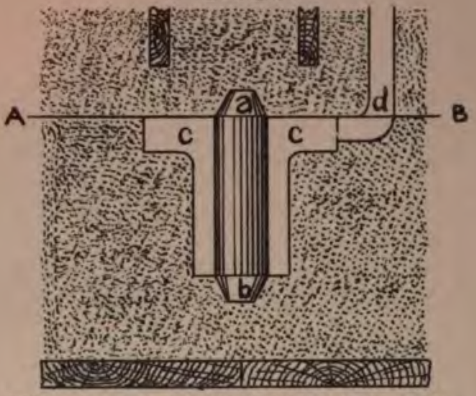
FIBROUS TEXTURE OF WROUGHT IRON.

FIG. 149.



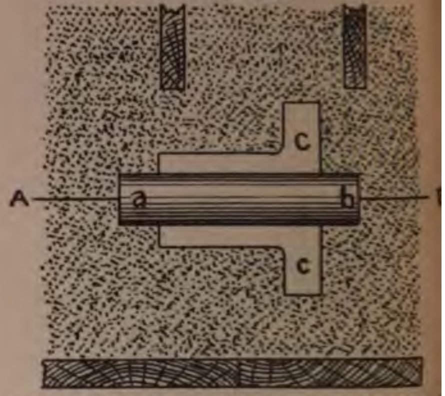
SNAP FLASK.

FIG. 151.



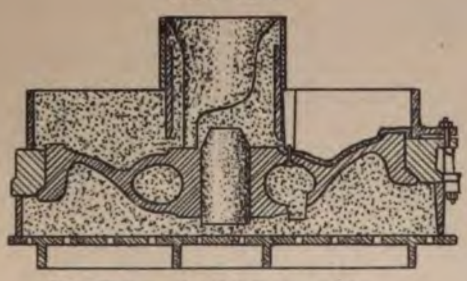
MOULDING A GLAND WITH VERTICAL, DRY SAND CORE.

FIG. 152



MOULDING A GLAND WITH HORIZ
TAL, DRY SAND CORE.

FIG. 155.



CASTING A CAR WHEEL.

FIG. 153

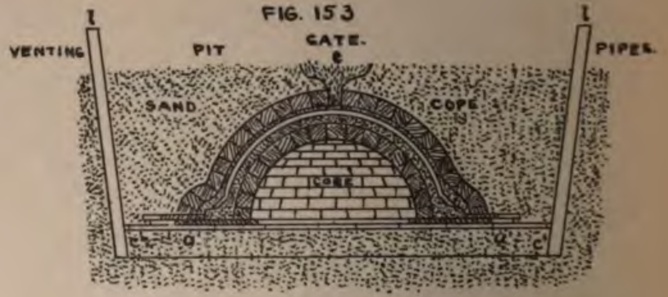
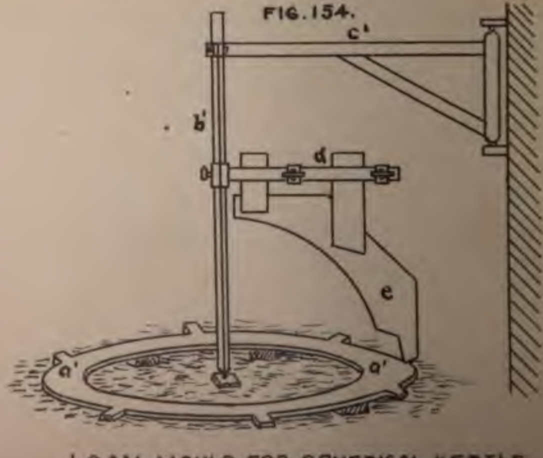
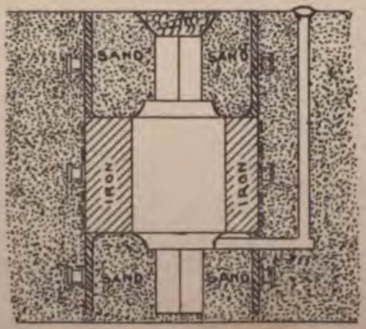


FIG. 154.



LOAM MOULD FOR SPHERICAL KETTLE.

FIG. 155A



CASTING A CHILLED ROLL.

ANALYSES OF MOLDING SAND.[1]

	IRON WORK.				BRASS WORK.	
	Light.	Medium.	Heavy.	General.	Light.	General.
SiO_2	82.21	85.85	88.40	75–85	78.86	. . .
Al_2O_3	9.48	8.27	6.30	5–13	7.89	. . .
Fe_2O_3	4.25	2.32	2.00	<5	5.45	<7
$CaCO_3$	0.68	0.29	. . .	<1.50	1.46	<2.25
MgO	0.32	0.81	0.50	. . .	1.18	. . .
Na_2O	0.09	0.10	. . .	<0.75	0.13	. . .
K_2O	0.05	0.03	. . .		0.09	. . .
MnO	. . .	Trace	0.25	. . .	Trace	. . .
Combined H_2O	2.64	1.68	1.73	<4	3.80	. . .
Organic matter	0.28	0.15	0.04	<1	0.64	<1
Specific gravity	2.652	2.645	2.630	. . .	2.64	. . .
Degree of fineness[1]	85.18	66.01	46.86	. . .	94.88	. . .

[1] See below.

Molding sand should possess some cohesiveness, be sharp, of uniform grain (open), refractory, and free from substances that give off gas when heated. The cohesiveness is necessary to permit ramming into the mold; it is caused by the presence of small amounts of clay substance. Sharpness of grain causes the single particles to adhere to one another; this is not the case with rounded particles. Uniformness of grain is desirable to a certain extent, in order to have sufficient capillary spaces through which steam and gases may pass off freely. According to the "Degree of Fineness," there are five grades of molding sand: superfine (> 100 per cent.), fine or light (90–100 per cent.), medium (75–90 per cent.), coarse or heavy (55–75 per cent.), extra coarse (30–55 per cent.). In determining the percentage of fineness, 100 grams sand are passed through a nest of five sieves (100-, 80-, 60-, 40-, and 20-mesh), the products weighed, and then the following calculation carried out: —

	Weight of sand, grams.		Passing through mesh screen.		Product.
	55.22	×	100	=	5,522.00
	20.89	×	80	=	1,671.20
	11.64	×	60	=	698.40
	10.57	×	40	=	422.80
	1.20	×	20	=	24.00
	0.06	×	1	=	0.06
	99.58				
Loss	0.42	×	60	=	25.20
					8,363.66

8,363.66 : 100 = 83.64 per cent. fineness.

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[1] Scott, "Iron Age," November 1, 1900.

The sand ought to be refractory, so as not to be affected by the hot iron. Any gases given off by the sand (CO_2 from $CaCO_3$) spoils the casting.

If in ramming the sand packs too tightly it can be loosened by adding old sand (the clay substance of which has lost its combined water, and with it its binding power), charcoal, coke, etc. If the sand does not pack satisfactorily its cohesiveness is increased by adding loam.

In molding,[1] the sand is rammed around the pattern in a box of wood or iron called flask.[2] Figure 148 represents a flask ready for casting a gland or cover of stuffing box. It consists of two parts, the lower drag *A*, resting upon the molding board *B*, and the upper cope *C*; both have handles, *D*, through which pass tie-rods, *t*, secured by nuts to give strength to the frame. In order to insure accurate and easy fitting, the cope has a dowel pin *a* and the drag a corresponding plate *b* with hole (see Figure 149). The cope has a number of crossbars, *j*, to prevent the sand from falling out when it is lifted from the drag. Wooden flasks are made of boards 1.5 inches thick; in iron flasks the casting is 0.5 inch thick and upward. The sand is from 2 to 4 inches thick on the sides and the bottom of the pattern. Snap flasks (Figure 149) are used when a large number of light castings has to be made from the same pattern; the flask is removed as soon as one mold is finished and used for another. This is permissible when the amount of metal poured is sufficiently small for the sand to hold it without being reënforced by the flask. In molding the gland, shown in Figure 148, the pattern and drag are placed on the molding board, as seen in Figure 150, and sand is sifted over the pattern until the mold is filled. Sand is rammed down in the cylindrical opening to form the core, then the sand around the pattern is tamped firmly and more added until the drag is filled. Any excess of sand is scraped off, a second molding board placed on the drag, and the two bound with clamps, of the form of channel iron, and wedges. The drag is inverted and the upper board removed, when it will be in the position shown in Figure 148. The surface is smoothed, sprinkled with sharp, dry sand (parting sand, SiO_2, 98.04; Al_2O_3, 1.40; Fe_2O_3, 0.06; CaO, 0.20; MgO, 0.16; combined H_2O, 0.14; specific gravity, 2.592),[3] the cope put in position, filled in the same manner as the drag (the sand being rammed around a stake 1.5 inches in diameter to form the inpour), the stake removed, the cope put to one side, the pouring gate enlarged, the parting sand brushed aside, and the groove *e* cut out which leads from *d* to *b*. Before taking out the pattern the sand surrounding it is slightly moistened to make it hold together better. Now a sharp iron (lifter) is driven into the pattern, rapped gently, and raised with the pattern. The mold is finally blackened (one part charcoal : one black lead : one Lehigh blacking), either wet with a swab, or dry from a linen bag, the cope placed on the drag and clamped to prevent the iron from lifting. After standing a while the mold is ready for pouring.

In most cases cores are made of dry sand (see below) instead of green sand. Figure 151 represents such a mold. The pattern, holding the wooden core cylinder with conical end *b* protruding, is placed in the drag and this filled with sand, turned, the loose core print *a* put in position, the cope placed on the drag and filled with sand. Now the pattern is removed, the separately prepared cylindrical dry-sand core is trimmed at *a* and *b*, placed and adjusted,

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[1] Hand and machine molding: "Engineering Magazine," 1903, XXIV, p. 695. Machine molding: "Iron Age," January 1, 1903.

[2] "Journal American Foundrymen's Association," IX, p. 24; "Iron Trade Review," August 29, 1901; "Stahl und Eisen," 1900, pp. 1041, 1098.

[3] Scott, "Iron Age," November 1, 1900.

and the cope replaced. If the pattern had to be molded horizontally instead of vertically, it would have to be split, as shown in Figure 152, one-half being molded in the drag, the other in the cope.

The casting temperature[1] with green-sand molds is low in order to prevent burning the sand (scabbing). The metal, hot from the cupola, is held in ladles until it has reached the right temperature. Dry sand and loam stand hotter metal than green sand, and with iron molds a very hot metal is desired.

When the casting is finished, toward the close of the day's work, the sand is removed from the heavier parts of a casting that these may cool at the same rate as the lighter parts. The rest of the sand is then shaken off and the sprues (leading from the gates to the molds) are knocked off with a hammer. The castings are now ready to be cleaned, which usually takes place the next morning. Cleaning consists in removing the adhering sand by rattling (small castings are charged into a horizontal revolving cast-iron cylinder, the tumbler),[2] by brushing with strong steel bristles (the common method), or sand blasting and chipping with compressed air (with large castings).[3]

Dry-sand Molds are made of a loamy sand, *i.e.*, one that runs high in SiO_2, low in Al_2O_3 (<5 per cent.) and Fe_2O_3 (<2.5 per cent.). The following analyses represent two extremes:[4] —

SiO_2	Al_2O_3	Fe_2O_3	$CaCO_3$	$CaSO_4$	MgO	$Na(K)_2O$	Combined H_2O	Organic matter.
94.30	1.95	0.33	1.63	. . .	0.54	0.05	1.05	0.15
69.31	4.76	1.58	3.50	8.19	7.77	0.12	2.95	1.82

On account of its clayey character dry sand is more binding and stronger than green sand, but also very dense, in fact too much so to allow steam or gases to escape in casting. The molded sand has therefore to be dried[5] and baked in an oven before a casting can be made. For this reason the molds are always made of iron. Baking drives off the moisture and organic matter, shrinks the sand, and thus causes it to become hard, strong, and sufficiently porous for the gases to escape. Dry-sand molds are used for most large and intricate castings; steel is always cast (bottom-casting) in dry sand. The dry sand used for cores is a sharp, pure sand, the grains of which are cemented together by mixing with bonds, such as beer, flour, molasses, glue, rosin, and clay. The cores when dry are baked in a manner similar to the molds.

Loam Molds. The loam (or sandy clay) is applied in a pasty condition to a brick or iron mold, faced with carbonaceous matter and dried. In order to make it dry evenly and to render it porous when it has been dried, it is mixed in a pug mill with organic materials, such as horse manure, cow hairs, chopped straw, tanning bark, etc. While with green- and dry-sand molds

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[1] Moldenke, "Transactions American Institute of Mining Engineers," February, 1904.

[2] See malleable castings below.

[3] "Compressed Air," VII, p. 2101.

[4] Scott, "Iron Age," November 1, 1900.

[5] "Stahl und Eisen," 1904, p. 585.

patterns are used which have the forms of the articles to be cast, with loam the molds are built up. Loam molds are used mainly with heavy, curved castings. The molding of a spherical kettle is shown in Figures 153 and 154.[1] On a cast-iron bed plate, *a′*, is erected a brick dome 4 inches thick, leaving out the keystone; loam serves as mortar. The dome is smeared over with a 2-inch layer of loam and smoothed off with the core board *e*. A fire is kindled in the dome and kept smoldering to dry out the brickwork and the loam cover. While drying, the cover is painted with a clay wash mixed with fine charcoal. When dry, the core is covered with a second layer of loam of the thickness of the casting, rounded off by another board attached to the spindle, painted and dried. On top of this come the third layer of loam, 2 inches thick, and lastly a course of 4-inch brick, to which the loam adheres firmly. This third layer with its brickwork forms the cope. The whole is ready to be well baked. After this the cope is lifted off with a crane, the second layer of loam broken and removed, the opening in the core closed, and the cope put again in place. The opening in the cope is left intact and serves as a gate for pouring. Painting the layers of loam where they come in contact with one another prevents their adhering. When the cope has been set, it is keyed to the drag, and the two are raised, lowered into a pit, and this filled by ramming with green sand. The mold is now ready to receive the metal.

Iron molds (chills) are used when the surface coming in contact with the iron is to be chilled. Thus, for example, in casting car wheels[2] in this country the surface is to be chilled to a depth of $\frac{3}{8}$–$\frac{6}{8}$ inch.[3] This is done by casting, as shown in Figure 155.[4] At *A* the rim is chilled by coming in contact with iron, while the rest of the wheel, being cast in sand, cools slowly. The composition of car-wheel iron[5] used to be given as having the following range: Gr. C., 2.75–3.00; C. C., 0.50–0.75; Si, 0.50–0.70; Mn, 0.50–0.75; S, 0.05–0.07; P, 0.35–0.45. At present this is narrowed down to Gr. C., 2.90; C. C., 0.60; Si, 0.70; Mn, 0.40; S, 0.08; P, 0.05.[6]

Drop tests of car wheels: "Foundry," 1901, XVIII, p. 59; "Proceedings American Society Testing Materials," III, p. 256. Photomicrographs: "Iron Age," February 20, 1902; "Proceedings American Society Testing Materials," II, p. 226.

SECTION 41. **Malleable Cast Iron.**[7] The manufacture of malleable castings, first men-

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[1] J. Sharp, "Modern Foundry Practice," London, 1900, p. 416.

[2] Car-wheel foundries: "Foundry," XIX, p. 140; XXII, p. 84; "Iron Trade Review," May 16, 1901; "Iron Age," May 16, 1901.

[3] Chill tests: "Journal Franklin Institute," 1900, CL, p. 351; "Proceedings American Society Testing Materials," 1902, II, p. 217.

[4] "Stahl und Eisen," 1895, p. 1050.

[5] Henderson, "Foundry," XV, p. 49.

[6] Specifications: "Transactions American Institute of Mining Engineers," February, 1904.

[7] Malleable foundries: "Foundry," 1902, XIX, p. 186; "Stahl und Eisen," 1903, p. 22; "Iron Age," February 26, 1903. History in United States: "Iron Age," October 13, 1898; February 9, 1899. General discussions: Forquignon, "Annales de Chimie et de Physique," ser. V, XXIII, p. 443 ("Stahl und Eisen," 1886, pp. 380, 777); Ledebur, "Stahl und Eisen," 1886, p. 383; 1897, p. 628; 1899, p. 366; 1902, p. 813. Stanford, "Transactions American Society Civil Engineers," 1895, XXXIV, p. 7 ("Iron Age," August 15, 1895); Wheeler, "Iron Age," February 13, 20, March 30, April 6, 1899; Davis, "Journal American Foundrymen's Association," 1900, VIII, pp. 93, 121; "Foundry," XVI, p. 103; "Engineering and Mining Journal," LXIV, p. 382; "Technology Quarterly," 1901, XIV, p. 60; James, "Journal Franklin Institute," 1900, CL, p. 227; Outerbridge, *op. cit.*, 1903, CLV, p. 289; "Proceedings American Society Testing Materials," 1902, II, p. 229; Wüst, "Stahl und Eisen," 1903, p. 1136; Moldenke, "Foundry," 1904, XXIII, p. 163; "Iron and Steel Magazine," 1904, I, p. 159. Photomicrographs: "Transactions American Institute of Mining Engineers," XXX, p. 734. **Specifications:** *op. cit.*, February, 1904.

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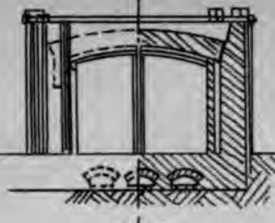
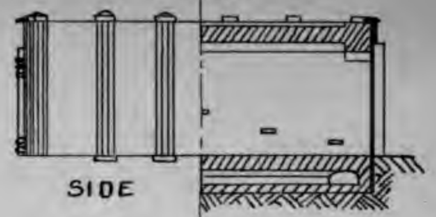
FIG. 161.



END

FIG. 160.



ANNEALING OVEN
FOR
MALLEABLE CAST IR

FIG. 159.

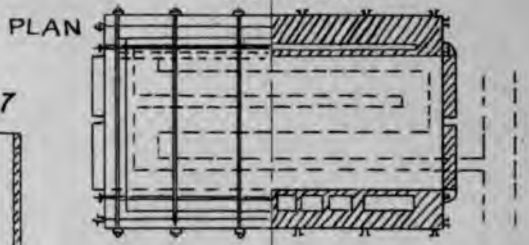
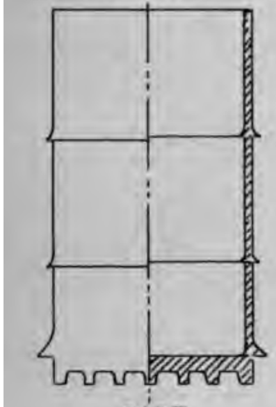
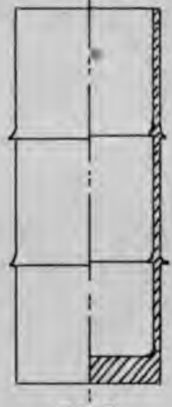


FIG. 156.



SIDE

FIG. 157



END

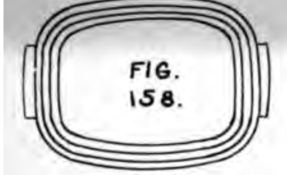


FIG. 158.

PLAN

ANNEALING POT
FOR
MALLEABLE
CAST IRON

FIG. 166.

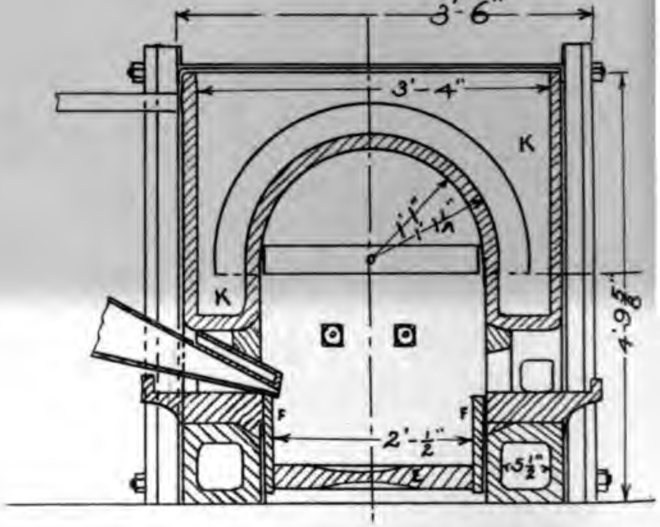
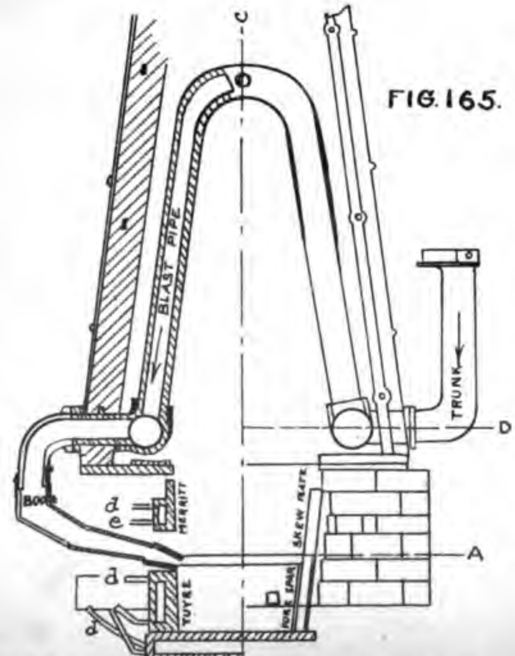
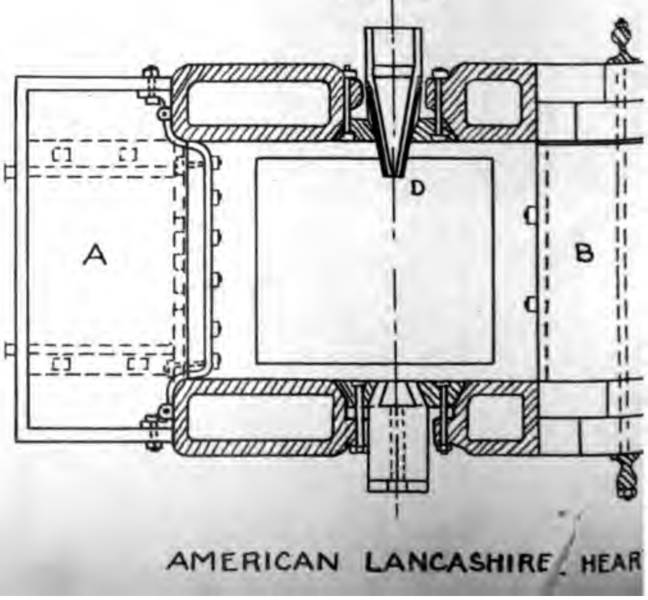


FIG. 165.



AMERICAN BLOOMARY FURNACE.

FIG. 167.



AMERICAN LANCASHIRE HEAR

tioned by Réaumur in 1722 and carried out today in the manner proposed by Lucas in 1804,[1] consists in decarburizing white iron by packing with oxide of iron in a pot, heating for several days in a furnace to a temperature of below 1,000° C., and cooling slowly. The surface, converted into a kind of wrought iron, can now be easily cut by tools. Malleable castings are used for elbows, tees, unions which have to be threaded; for handles, latches, and similar small hardware; for harness mountings, plowshares, iron handles, small parts of machinery, etc.

In heating carburized iron with oxidizing reagents, only combined carbon is oxidized, graphitic carbon being removed with great difficulty; hence the raw material must be a white iron. Under annealing conditions, hardening carbon (martensite) is changed first into carbide carbon (cementite), and this breaks down into finely-divided temper carbon and iron (ferrite). The C is oxidized in the pot by the oxide of iron to CO, and this on leaving the pot by the O of the air to CO_2. During the heat, while the carbon on the surface is being oxidized, part of the carbon in the centre diffuses outwardly, and is oxidized. Thus it is possible to decarburize castings 1 inch in thickness. Thicker castings are toughened by the process by being decarburized to a certain depth if the heating be prolonged sufficiently; even gray castings are benefited by the treatment. Roylston,[2] in examining annealed bars, could distinguish four layers; the outer layer consisted of crystals of ferrite, between which were discernible particles of crystallized silica (Si from pig iron oxidized on the surface); then came ferrite and pearlite, then pearlite, then cementite and ferrite, the latter impregnated with graphite. Child and Heineken[3] have shown by analytical and microscopical examinations that the breaking down of the carbide takes but a short time. Photomicrographs prepared by them show clearly the different changes that take place.

Mn retards the decarburization and increases the shrinkage; it should not exceed 0.4 per cent., usually varies between 0.1 and 0.2 per cent. P is indifferent, but on account of resilience should not exceed 0.225 per cent. S retards the decarburization and should not exceed 0.05 per cent.; *e.g.*, there were required for annealing, in the presence of 0.150 per cent. S, three and one-half days; of 0.200 per cent., five days; and of 0.250 per cent., nine days.

As white iron shrinks considerably more than gray iron, and is liable to give rough castings, it is necessary to have 0.4–0.8 per cent. Si, and to keep the total carbon below 3 per cent., so that no graphitic carbon may separate out. The thicker the casting the lower the Si; for ordinary work Si averages 0.65 per cent., for heavy work, 0.45 per cent. According to Ledebur,[4] Si may reach 1 per cent. See in this connection Charpy-Grenet in the "Metallographist," 1902, V, p. 202. Carbon should be over 2.75 per cent. The tensile strength of malleable castings lies between 42,000 and 47,000 pounds; castings showing 35,000 pounds are serviceable for ordinary work. The elongation ranges between 2½ and 5½ per cent.; the deflection of a bar 1 inch square resting upon supports 12 inches apart should be over ½ inch, with a breaking weight of 35,000 pounds. It was formerly believed that only charcoal pig and scrap (sprues, gate-runners, shrinkers) were suited for making malleable castings. Coke iron, how-

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[1] Beck, "Geschichte des Eisens," IV, p. 109.

[2] "Journal Iron and Steel Institute," 1897, I, p. 160.

[3] "Transactions American Institute of Mining Engineers," XXX, p. 734.

[4] "Stahl und Eisen," 1902, p. 813.

ever, works just as satisfactorily, as long as it comes up to the chemical requirements stated above.[1]

The iron is melted in the cupola or the reverberatory furnace, and cast in flasks.[2] Open-hearth furnaces have been introduced at a few plants. A 10-ton open-hearth furnace will make three heats a day, and require 1 ton coal to 6 tons of iron; while with direct firing, 1 ton coal to 4 tons of iron represents excellent work, the more common figure being 1 ton coal to 2 tons iron. Heats larger than 15–18 tons are not desirable, as the metal is liable to become overheated and oxidized. An open-hearth furnace makes 300 heats before any repairs are needed, and 1,000 heats before rebuilding becomes necessary.[3]

The castings are chipped to remove fins, sorted, and then packed in annealing pots with clean, heavy forge scales. A pot made in three sections is shown in Figures 156–158.[4] It weighs 750 pounds and holds 800 pounds iron; it weighs when loaded about 1 ton, and is handled by six men with a carriage. It lasts five heats of five days each; in each heat some scale is made which, peeling off or being knocked off, shortens the life. The annealing oven[5] shown in Figures 159–161,[6] 20 feet long x 8 feet wide x 5 feet 6 inches high to spring of arch, holds thirty of the above pots. It is heated with oil, the burners being placed at either end of a side wall. The flame passes through the fire holes into the furnace, in which the annealing pots have been stacked, leaving spaces open between them. It leaves the furnace through openings in the opposite wall, descending in a vertical flue, zigzags in bottom flues, and then passes off into the chimney. The temperature is usually a cherry red; a higher temperature is of no advantage; it simply shortens the life of the pots. Roylston[7] gives 860–900° C. as a common range. Moldenke[8] states that the lowest permissible temperature is 680° C., that a safe temperature is 780° C., and that cupola metal requires 850° C. In general it may be said that a higher temperature for a shorter time has a similar effect as a lower temperature for a longer time; the latter is, however, preferable, as the oxidation of C is more satisfactory. The time for annealing is from three to ten days, according to size of the furnace and of the castings. The fuel ought not to be burned completely in the fire hole, but the combustion ought to be finished in the oven. Uniform temperature is a necessity. The cooling of the oven takes one or more days. The oven described above consumes 100 gallons oil per burner per day, air being used as atomizer. Usually malleable founders charge a furnace once a week. They fire up quickly, *i. e.*, thirty-six hours or more, maintain the temperature for at least two days, and then cool down slowly, opening the furnace only when the pots have reached a black heat.

Freshly annealed iron has a rich blue color due to a thin scale, which is worn off when the iron is cleaned. This is done by dry tumbling, when the appearance of the surface is of little

[1] Wheeler, "Iron Age," March 30, April 6, 1899.

[2] Casting temperature: "Journal Iron and Steel Institute," 1903, I, p. 457.

[3] Moldenke, "Foundry," 1904, XXIII, p. 163.

[4] "Transactions American Society of Civil Engineers," 1895, XXXIV, p. 7.

[5] See also "Iron Age," February 26, 1903.

[6] "Transactions American Society of Civil Engineers," 1895, XXXIV, p. 8.

[7] "Journal Iron and Steel Institute," 1897, I, p. 163.

[8] "Foundry," 1904, XXXIII, p. 163.

importance. A dry tumbler is a horizontal revolving barrel (from 16 inches diameter x 28 inches length to 42 inches diameter x 72 inches length), into which the castings are charged. If the casting is to show a smooth surface, dry tumbling is followed by wet tumbling (water tumbler, from 16 inches diameter x 28 inches length to 20 inches diameter x 32 inches length), and this by dry polishing (tumbling with pieces of leather).

The following table shows clearly the chemical changes that take place in annealing, and gives the mechanical properties of the malleable castings.

AVERAGE OF FIFTEEN TESTS.

Material.	T. C.	C. C.	Gr. C.	Mn	Si	P	S	Hours annealed.	Loss in C	Tensile strength, pounds per square inch.	Reduction of area, per cent.	Elongation, per cent.
Sprues	3.14	2.93	0.21	0.26	0.64	0.181	0.046	. . .	. . .	. . .	. . .	. . .
Annealed iron . .	2.45	0.23	2.22	0.26	0.64	0.181	0.046	119	0.69	49,100	4.2	5.2
Sprues	3.04	2.85	0.19	0.21	0.73	0.154	0.050	. . .	. . .	. . .	. . .	. . .
Annealed iron . .	2.66	0.31	2.35	0.21	0.72	0.153	0.050	119	0.38	49,810	5.23	6.61
Skin	1.34	0.61	0.73	0.15	0.65	0.154	0.116	120	. . .	. . .	. . .	. . .
Interior	3.51	0.02	3.49	0.14	0.63	0.158	0.113		. . .	. . .	. . .	. . .

The cost of annealing in the furnace described (thirty pots at 800 pounds iron, two oil burners) was, per pound of annealed iron: —

Labor, packing pots	0.0117 cents
Labor, loading oven	0.0025 cents
Fuel (including air for atomizing and attendance)	0.0938 cents
Labor, unloading oven	0.0025 cents
Labor, dumping pots and picking out iron	0.0104 cents
Labor, tempering and handling packing pots	0.0083 cents
Pots	0.1000 cents
Scale for packing	0.0208 cents
	0.2500 cents

According to Moldenke,[1] this country produces annually about 650,000 tons malleable castings.

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[1] "Foundry," 1904, XXXIII, p. 163.

CHAPTER IV.

PRODUCTION OF WROUGHT IRON.

SECTION 42. **Introductory.** Wrought Iron or Weld Iron (see page 2) is commercially pure iron which, having been aggregated from pasty particles without subsequent fusion, contains more or less intermingled slag. When a ball of wrought iron is being squeezed, hammered, or rolled at a welding heat into a bar or a rod, the single granules of iron, each surrounded by a film of slag, become elongated. This gives wrought iron the characteristic fibrous texture which is seen especially in the fracture when a bar is nicked and bent upon itself (Figure 162). A microscopical section (Figures 7 and 163) will show ferrite (perhaps a small amount of pearlite), and here and there a film of slag. Wrought iron contains about 2 per cent. slag. The amount will be greater when a large ball is being produced than when a small one; when the temperature is low than when high; when the mechanical treatment is restricted (squeezing out the slag) than when extended. The composition of the slag is often the same as that produced in the furnace, but it is richer in iron in case the granules have been exposed to strongly oxidizing influences, *e.g.*, in puddling. Ledebur,[1] *e.g.*, gives as an example for one puddled bar 2.34 per cent. slag with 58.40 per cent. Fe (as FeO and Fe_2O_3), 0.40 per cent. SiO_2, 16.10 per cent. P_2O_5; for another, 0.41 per cent. slag with Fe, 28.56; SiO_2, 15.18; P_2O_5, 26.01; Mn, 0.80; CaO, 3.93; Al_2O_3, 13.17. Wrought iron melts at about 1,600° C. At a white heat it is pasty and readily welded. It is tough and malleable, ductile and forgable, especially at a red heat. On account of its low percentage of C (under 0.1 per cent.), it is soft, and does not harden upon quenching. The tensile strength (Lanza) should be over 45,000–46,000 pounds, the limit of elasticity as much as 27,000–28,000 pounds, the contraction of area at fracture at least 30 per cent. It should also stand bending double cold, red-hot, and at a flanging heat.[2]

Up to the middle of the present century nearly all malleable iron produced was wrought iron; since then it has been mostly replaced by low-carbon steel. There are, however, many uses for which it still holds its own.

Classification of Processes: —

I. Direct Production of Wrought Iron: in hearth furnaces (Catalan and American Bloomery Process); low-shaft furnaces (Osmund Furnace); high-shaft furnaces (Husgafvel High Bloomery Furnace); retorts (Blair Process); and reverberatory furnaces (Siemens Process, Eames Process).

II. Indirect Production of Wrought Iron: in hearth furnaces (Walloon, Lancashire, South Wales Process, etc.); in reverberatory furnaces (Dry and Wet puddling).

The direct processes, starting with ore, were formerly exclusively used. They have had to yield, with a few exceptions, to the indirect processes, which start in with pig iron as their raw material.

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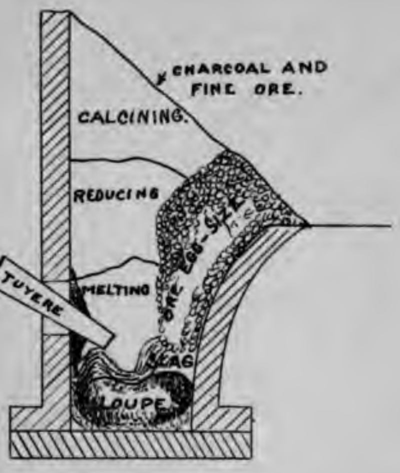
[1] "Handbuch," 4th ed., p. 862.

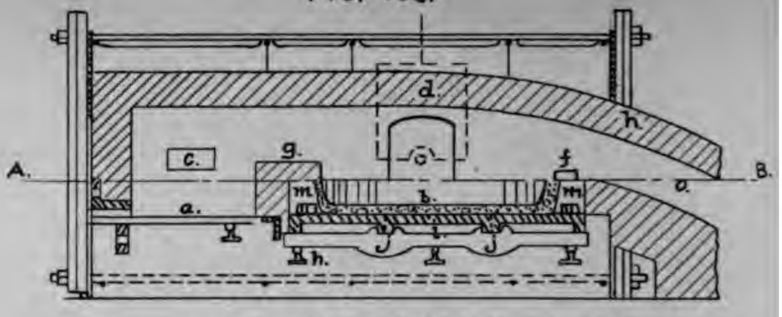
[2] Holley, "Transactions American Institute of Mining Engineers," VI, p. 101.

FIG. 164.



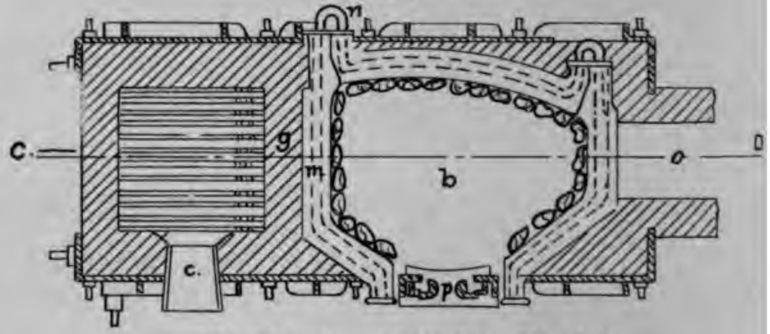
CATALAN FORGE.

FIG. 168.



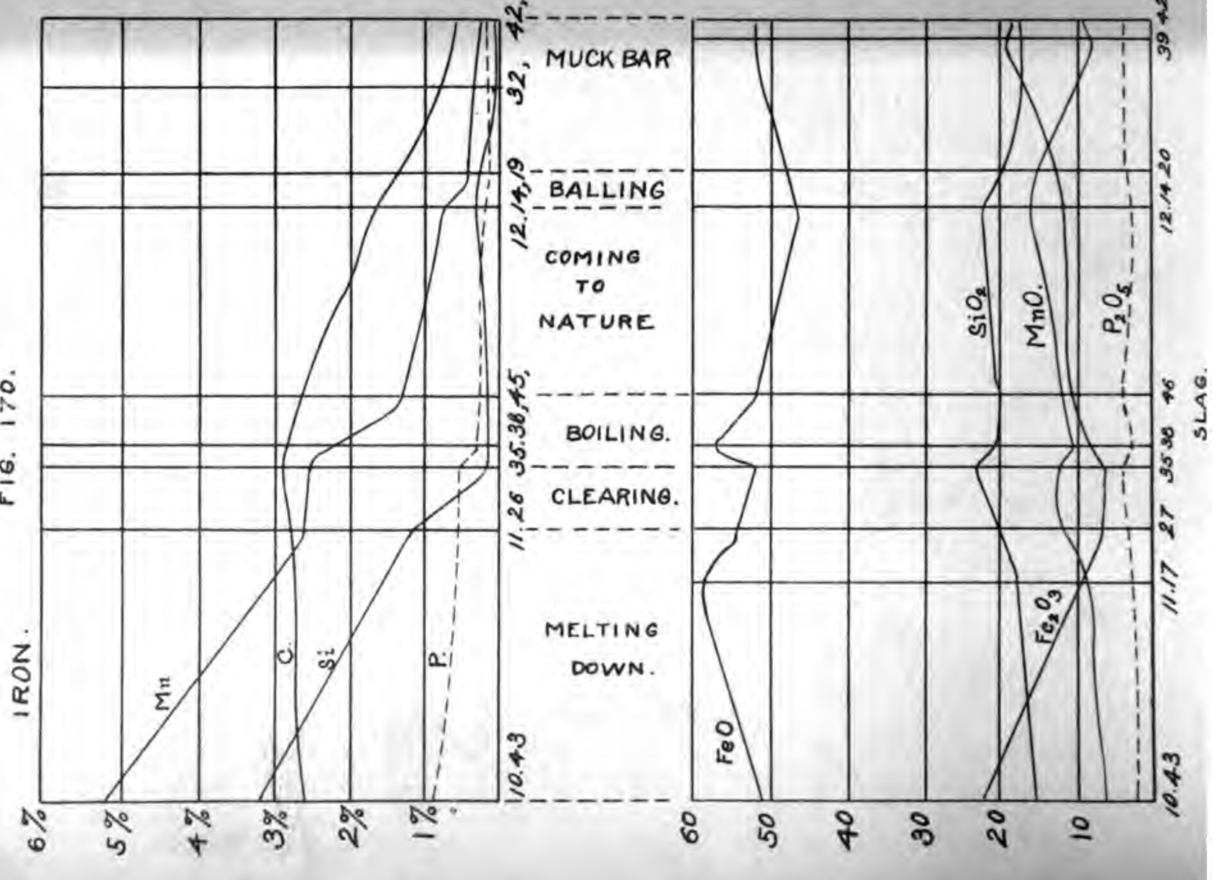
LONGITUDINAL SECTION ON C D.

FIG. 169.



HORIZONTAL SECTION ON A B.
PUDDLING FURNACE.

FIG. 170.



SECTION 43. **Direct Production of Wrought Iron.**[1] In direct processes, wrought iron is produced from ore in one operation with solid or gaseous fuel at a low heat (to avoid carburization) and in the presence of a ferruginous slag (to hold the phosphorus in solution). The iron, which is obtained in the form of a spongy mass mixed with slag (the slag filling the cells of the sponge), is subjected to mechanical treatment (squeezers, hammers, rolls, etc.) which removes the large excess of slag and gives the wrought iron its marketable form. The processes require rich and cheap ores, much fuel low in or free from S, and power; the loss of iron is large (over 20 per cent.), the production is small and intermittent, the product is not uniform.

(1) *The Catalan Forge* (Figure 164)[2] is a low, one-tuyère hearth from 20 x 20 and 16 inches deep to 40 x 32 and 27 inches deep; the tuyère is inclined downward 30–40°, terminating from 9 to 15 inches above the bottom. The hearth is lined with heavy iron plates, and receives a charge weighing 300–900 pounds of ore. Supposing the hearth to be red-hot from a previous charge, it is filled with charcoal to the level of the tuyère, then ore of nut size is charged toward the front or working side, and charcoal toward the back, or tuyère side, the two vertical columns being kept apart during charging by a piece of sheet iron. The inclined upper face of the ore is covered with moistened charcoal fines to prevent the CO from passing and to force it through the ore. Blast is now turned on, the hot CO passes off mainly through the loose, coarse ore and begins to reduce it; some gas, of course, rises through the fuel column and burns with a blue flame to CO_2. As the charcoal burns away, it is replenished; layers of fine ore are added; they are moistened to prevent the ore from being blown away and from trickling downward. In the process, at first a basic iron slag rich in P forms[3] and, collecting on the bottom of the hearth, is tapped. About two hours after starting, the ore is slowly pushed down into the hot region of the tuyères as successive layers become sufficiently reduced by the action of the CO; the reduction is completed by solid C at a white heat. Unreduced parts of the ore combine with the gangue to a slag which surrounds and permeates the metallic iron which has become pasty. When all the ore has been reduced the blast is stopped, the charcoal drawn aside, the ball of iron ("loup") pried up with bars, removed with heavy tongs, and brought to a tilting hammer. A charge of 1,000 pounds limonite is worked in about six hours. For 100 pounds of iron there are required 300–320 pounds ore and 340 pounds charcoal.

The *Reynolds* Process[4] resembles in practice and product the Catalan forge, only pulverized bituminous coal has replaced charcoal and a rotary furnace the primitive hearth.

(2) *The American Bloomery Process*[5] was extensively carried out in New York and New Jersey. It resembles the old German bloomery process in that fine ore and charcoal are intimately mixed and charged into the hearth. The furnace is shown in Figure 165.[6] The hearth, 20–30 inches wide, 27–32 inches long, and 23–40 inches deep, is lined with heavy cast-

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[1] Howe, "Metallurgy of Steel," p. 259; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 129; Ehrenwerth, "Stahl und Eisen," 1891, pp. 299, 727, 978; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1891, pp. 35, 359, 456, 545; 1892, pp. 41, 53. Leobner, "Oesterreichisches Jahrbuch," 1902, p. 1; Jacobsson, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1902, p. 105.

[2] Percy, "Iron and Steel," p. 300.

[3] Stead, "Journal Iron and Steel Institute," 1877, II, p. 372.

[4] "Bulletin American Iron and Steel Association," March 10, 1904; "Iron Trade Review," March 31, 1904; "Iron and Steel Magazine," 1904, I, p. 491.

[5] Egleston, "Transactions American Institute of Mining Engineers," VIII, p. 515.

[6] Howe, "Steel," p. 270.

iron plates. It has a ⬬-shaped tuyère 1 x 1¾ inches and from 8 to 15½ inches above the bottom; the blast, superheated to 285–450° C. in overhead cast-iron pipes, has a pressure of 1½–2½ pounds per square inch. The hearth, filled with glowing charcoal, receives at intervals of from five to twenty-five minutes a mixture of charcoal, coarsely pulverized (washed) ore, and slag from a previous charge. The ore being reduced sinks through the fuel bed and collects in the form of a pasty ball of iron on the bottom, while the slag formed is tapped when necessary. After about three hours, the iron loup weighing 300–400 pounds is pried up, raised toward the tuyère level to be brought to a welding heat, withdrawn, hammered to a bloom, reheated in the hearth, and rehammered. In twenty-four hours are produced eight loups; there are required per short ton of blooms 250–300 bushels charcoal and 1.25–1.5 days' labor. The yield is 80 per cent. on the iron in the ore.

(3) *The Osmund Furnace*[1] is a low-shaft furnace 8 feet high. It stands between the hearth furnace (low bloomery) and the high-shaft furnace (high bloomery, "Stückofen"), which is about 16 feet high. Both work intermittently.

(4) *The Husgafvel High Bloomery*[2] is a shaft furnace 26 feet high, 5-foot bosh, and 4-foot throat, with a detached hearth. The shaft is a wrought-iron air jacket which superheats the blast passing through it to 150–250° C. The bosh, being exposed to a very high heat and to much wear, is made of cast-iron sections and readily exchanged. The hearth, a cast-iron box on wheels, stands on a lifting platform. It has a false bottom, to prevent the bloom from adhering to the true bottom, and two pairs of water-cooled tuyères, one opening near the bottom and the other near the top of the hearth. The furnace works continuously. Charcoal and ore are charged in horizontal layers and descend in the shaft in the usual manner. With a fresh hearth, the blast enters through the lower tuyère holes; after a while these are closed and the pipes are inserted through the upper holes. Enough slag is kept in the hearth to cover the forming bloom; any excess is tapped. When the bloom about fills the crucible, the blast is stopped, the hearth lowered and replaced by a new one filled with charcoal; the contents of the crucible is dumped onto a truck and the bloom worked. The blast, and thereby the process, is interrupted only about five minutes. The furnace produces in twenty-four hours 1½–3 tons of blooms, weighing 670–795 pounds. The loss in iron is 21–23 per cent. The process is in use in Finland.

(5) *The Blair Process*,[3] abandoned now, consisted in reducing by means of charcoal in vertical retorts heated with producer gas iron ore mixed with lime. The estimated cost[4] of the iron sponge to be used in the open-hearth steel process was $14.40 per ton. The process was an attempt to improve that of Chenot,[5] based on similar principles. Horizontal retorts[6] also proved unsuccessful. In the *Blair-Adams Process*,[7] tried at Pittsburgh, reducing gas entered the externally heated vertical retort, filled with ore mixed with 15 per cent. coal, to assist in the reduction; the reduced iron was transferred directly to a Siemens open-hearth furnace.[8]

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[1] Percy, "Iron and Steel," p. 320.

[2] Howe, "Steel," p. 271; Garrison, "Transactions American Institute of Mining Engineers," XVI, p. 334.

[3] Howe, "Steel," p. 278; Wedding, "Das Schmiedbare Eisen," 1875, p. 556.

[4] Blair, "Transactions American Institute of Mining Engineers," II, p. 195.

[5] Percy, "Iron and Steel," pp. 335–345.

[6] "Iron Age," XLII, p. 119; "Journal Iron and Steel Institute," 1889, I, p. 328.

[7] Howe, "Steel," p. 380; "Journal Iron and Steel Institute," American volume, 1900, p. 317.

[8] "Journal Iron and Steel Institute," 1890, II, p. 766.

(6) *The Siemens Direct Process,*[1] sometimes called "Precipitation Process," also abandoned, consisted in heating fine ore mixed with coal in a reverberatory furnace, the hearth of which was a revolving horizontal cylinder. The furnace was heated with producer gas and had regenerative chambers. The resulting metal was balled, squeezed to remove the slag, and then used either in the open-hearth process or rolled into bar iron.

(7) *The Eames or Carbon Iron Company's Process*[2] was carried out for some time at Pittsburgh, but has also become obsolete. It consisted in reducing iron ore mixed with retarded coke (ground coke or coke dust mixed with cream of lime to prevent rapid combustion) in a long reverberatory furnace fired at both ends with natural gas. A charge of 2,240 pounds of rich ore and 550 pounds retarded coke was worked in three and one-fourth hours and yielded 1,550 pounds sponge for the open-hearth furnace, or 1,160 pounds squeezed blooms (Fe, 98.47; P, 0.02; S, 0.021; SiO_2, 0.94; C. C., 0.150), or 1,000 pounds muck bar (Fe, 99.02; P, 0.01; S, 0.015; SiO_2, 0.56; C. C., 0.100).

SECTION 44. **Indirect Production of Wrought Iron in General.** In this method of working, which starts with pig iron as raw material, two processes, as shown above, may be distinguished: The Charcoal Hearth Processes, in which the Si, Mn, and C of the pig iron are oxidized by the action of the blast (impinging upon the drops of the iron as it is being melted down) and by the action of the basic ferruginous slag. This melting down is repeated until a product of the desired quality is obtained. The resulting wrought iron is, therefore, homogeneous and tough. As the elimination of P is imperfect, only pig iron low in P can be used. If the operation be carried only far enough to remove Si, or Si and Mn, there will result a refined cast iron (refinery furnace). In the Puddling Processes, carried on in reverberatory furnaces, the Si, Mn, C, and P are removed by the oxidizing action of the oxide of iron charged and the basic ferruginous slag formed; the product is not so uniform as above, but the elimination of P is more perfect.

SECTION 45. **Charcoal Hearth Processes.**[3] According to the number of times that the metal is melted down, the processes are classed as: —

Single Melting or Slag-bottom Process, Double Melting (Swedish Walloon, Lancashire, and South Wales Processes), and Triple Melting (German and Franche-Comté Processes).

(1) *The Slag-bottom Process.*[4] The method is practiced in Styria, Austria. White iron, in pieces 7 or 8 inches wide and 1½ inches thick, is melted down in a small charcoal hearth, the bottom of which is composed of rich finery cinder rammed down firmly. The iron is pushed toward the tuyère as fast as the hotter end melts down. While the iron trickles past the tuyère, it is decarburized and collects as a ball on the bottom. Imperfectly refined parts are treated over and over again. A charge of iron is about 230 pounds; the ball is divided into eight blooms, which are hammered. The blooms, before reheating, are immersed in molten cinder.

(2) *The Swedish Walloon Process* (called in Sweden plain Walloon).[5] This process is used exclusively in Sweden for producing from Danemora pig iron wrought iron, which is shipped to England, especially Sheffield, to be converted into blister steel, used for making

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[1] Howe, "Steel," p. 285; Wedding, "Das Schmiedbare Eisen," p. 269.

[2] Howe, "Steel," p. 374.

[3] Howe, *op. cit.*, p. 289.

[4] Howe, *op. cit.*, p. 293; Percy, "Iron and Steel," p. 610; Turner, "Iron," p. 264.

[5] Percy, "Iron and Steel," p. 599.

fine grades of steel. The hearth, which has the form of an inverted truncated pyramid about 30 inches square, is composed of cast-iron plates. It has two tuyères at the back, which are placed side by side about 7 inches above the bottom and fifteen beneath the top of the hearth. One or two pigs of mottled (perhaps white) iron (15–18 inches long, 9 inches wide, 3–4 inches thick at one end, 1–2 inches at the other) are melted down in the charcoal fire, being pushed forward as fast as the ends melt off, until about 100 pounds metal have been charged. The iron melting down, drop by drop, becomes decarburized in passing through the blast region and collects in a pasty mass on the bottom; the slag formed is tapped off at intervals. The iron, which is partly refined, is rolled up into a lump, raised to the top of the charcoal, melted down with the addition of rich slag and hammer scale, which completes the refining. The metal is balled to a bloom, reheated in the hearth, and sent to the hammer. The product is not very uniform, because the iron that is first melted is exposed for a longer time to the blast than that which is melted later. For 2,000 pounds of blooms there are required 4,250 pounds charcoal and 2,640 pounds pig iron; the loss in iron is 24 per cent.; the capacity of a forge is 3,900 pounds blooms in twenty-four hours.[1]

(3) *The Lancashire Process.*[2] Nearly all the charcoal hearth iron of Sweden, excepting the Danemora iron, is made by the Lancashire process. The process is also used in this country. The hearth is shown in vertical and horizontal sections in Figures 166–167. Legend: *A*, ore- or working-plate; *B*, shelf for preheating cast iron; *C*, hearth proper; *D*, water-cooled tuyère; *E*, water-cooled cast-iron bottom plate; *F*, cast-iron side plates; *I*, cast-iron water-cooled boshes; *K*, cast-iron water-cooled roof and sides. While the preceding charge is being worked, 275 pounds pig iron are placed on the shelf *B* and preheated. When the ball has been drawn, the hearth cleaned, and, if necessary, some slag added, the hearth is filled with charcoal, the red-hot pigs are drawn onto it, covered with charcoal, and the blast is let on. The iron while melting down is raised every few minutes so that it may not sink below the tuyère, but that the drops shall pass the oxidizing zone of the tuyère region and be decarburized. The pasty mass collecting on the bottom is broken up with bars, mixed up with decarburizing slags, raised above the tuyères, and, after being melted down, collects in a ball which is drawn and hammered. For 2,000 pounds blooms there are required 947 pounds charcoal and 2,300 pounds pig. The loss of iron is 13 per cent.; the capacity 5,300 pounds blooms in twenty-four hours.

(4) *The South Wales Process*,[3] used formerly in South Wales for making plates for tinning, has become practically obsolete, the metal having been replaced by Bessemer and open-hearth steel. The pig iron is melted down in a coke refinery,[4] where it loses part of its Si and C; it is then tapped into two charcoal refineries, where it is worked in a manner similar to that of the Lancashire process, excepting that the ball of iron is drawn without a final melt.

(5) *The German Process* starts with gray (perhaps mottled) iron. This is melted down in a charcoal hearth, which makes the iron white and decarburizes part of it. It is now raised as a whole or in part above the level of the tuyère and melted down twice, when the resulting ball is drawn and hammered. The *Franche-Comté Process* is similar.

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[1] Howe, "Steel," pp. 292, 295.

[2] Howe, *op. cit.*, p. 289; "Iron Age," July 16, 1903, p. 28.

[3] Howe, "Steel," pp. 293, 295.

[4] Percy, "Iron and Steel," p. 584.

SECTION 46. **Puddling.**[1] The increased cost of charcoal in the hearth processes induced Henry Cort, in 1784, to make wrought iron in a reverberatory furnace heated with mineral fuel. In his process, called Dry Puddling, white iron was melted down in a furnace having a sand bottom, and stirred so as to expose new surfaces to the oxidizing action of the air. The puddling was called dry, as the white iron never became very fluid and the amount of slag formed was comparatively small. As the loss in iron was very large, Baldwin Roger, in 1818, replaced the acid hearth by one of cast iron which greatly reduced the amount of slagged iron. About 1830 Joseph Hall lined the cast-iron hearth with oxidized iron, thus laying the foundation of modern Wet Puddling (pig boiling), in which the Si, Mn, C, and P are oxidized mainly by the solid oxygen of the red hematite ore which forms the lining of the furnace and makes a large amount of fluid cinder. At first only wrought iron was made; between 1840 and 1850 steel began to be produced in the furnace. Now the manufacture of steel by puddling is obsolete, and since low-carbon, open-hearth steel has replaced wrought iron in so many industrial uses, puddling has lost its former importance; but it still holds its own along certain lines where wrought iron is indispensable.

(1) *The Furnace.* The furnace shown in horizontal and vertical section in Figures 168–169[2] is a single-hearth reverberatory furnace fired with coal or natural gas. It has a large grate *a* and a small hearth *b* so as to permit obtaining quickly the intense heat necessary for the process. The fire hole *c* is open; it is closed by piling up coal. As it is essential to reach with the rabble every part of the hearth from the hole *e* in the working door *d*, the hearth has the form indicated in the plan. In order to weaken the cooling action of the air rushing in through the working door, the centre of the flue-bridge *f* is not placed opposite that of the fire-bridge *g*, but nearer the working side or front, say from 3 to 4 inches; further, the working door is placed nearer the fire-bridge than the flue-bridge, which facilitates the travel of the flame from the working door to the flue; and lastly, the roof *h* is made to slope from front to back. The flame, issuing forth from the fireplace, is thus drawn toward the front, where it can develop freely, and opposes the entrance of an excess of air. The bottom of the hearth is composed of three cast-iron plates, *i*, resting on bearers, *j*, supported by rails, *k*. The sides and the back consist of two water blocks, *m*, joined at the back by piping *n*, the water entering on the fire-bridge side. As a rule, only the ends are water-cooled. The flue *o* is usually inclined, that any slag formed may run off and be tapped. The heat of the flame issuing from the furnace is generally utilized to raise steam, as a rule in vertical (Cahall, Hyde, Moore) boilers.[3] If this is not the case, the gases pass off into the open trough, a separate stack with damper. Beneath the working door is the slag tap *p*. The furnace is enclosed in iron plates and well secured by buck-stays and tie-rods. In order to protect the workman against the heat radiated from the door *d*, a sheet-iron traveling water-cooled sliding door (not shown) is often suspended by rollers in front of it.[4] The inside dimensions of the furnace in common use are: hearth — length,

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[1] Turner, "Iron," I, p. 281; Louis, "Journal Iron and Steel Institute," 1879, I, p. 219; Head, *op. cit.*, 1893, I, p. 125; Roberts, "Transactions American Institute of Mining Engineers," VIII, p. 355. General results: "Zeitsch. d. Vereins deutsch. Ingenieure," 1872, XVI, p. 673; Cubillo, "South Staffordshire Iron and Steel Institute," November 28, 1900; "Colliery Guardian," November 30, 1900; "Stahl und Eisen," 1901, p. 404; Walker, "Iron Trade Review," January 10, 1901, p. 11.

[2] Ledebur, "Handbuch," 4th ed., p. 886.

[3] Mounting of Babcock and Wilcox boiler over puddling furnace: "Iron Age," April 6, 1893.

[4] "Iron Age," November 6, 1890; "Stahl und Eisen," 1890, p. 613, drawings.

6 feet; width at fire-bridge, 4 feet; at door, 4 feet 6 inches; at flue, 3 feet 6 inches; grate, 2 x 4 feet and 18 inches deep; top of fire-bridge to roof, 18 inches; top of flue-bridge to roof, 12 inches; inclined flue leading to near bottom of stack, 12 x 12 inches and 3 feet long; stack, 24 x 24 inches and 39 feet high. The life of a furnace is about one year, when it has to be completely overhauled.

(2) *Fettling or Fix* is the oxidizing material with which the hearth is lined (fettled or fixed). In putting in the working bottom the bottom plates are coated with clay and then covered with a layer of puddle cinder or mill cinder and scrap iron. This is melted down, hammer scale stirred in, and the whole patted into the form of a shallow basin, with a bottom 4–6 inches thick at the tap hole. The sides are now lined with blocks of hematite or magnetite, and the spaces between the individual blocks and between the blocks and the iron casing rammed full with fine iron ore ground in a wet pan to make a plastic mass (puddler's mine). The fine fettling has to be repaired after every heat, the lump fettling after every shift (about 1,200 pounds fettling for every ten heats), and the bottom after every week, scrap iron being commonly used.

(3) *The Pig Iron.* As the object of puddling is to remove Si, Mn, C, (S), P, etc., from the cast iron, the less these elements are present the more suitable the iron is for puddling. Gray forge or mottled iron form the usual raw material, although gray irons high in Si and white irons high in Mn are used for want of better material, or if the process is to be purposely retarded, *e.g.*, in puddling for steel. Of the harmful elements, S and P have to be especially considered. The longer it takes to puddle the iron the more thoroughly will the S be removed. With P, the more basic the slag and the longer that it is permitted to act upon the iron the more thorough will be its removal.

(4) *Mode of Operating.* Supposing the hearth to have been cooled down and repaired from a previous charge, 500–550 pounds pig iron in 1½-foot lengths are charged, and with them 200 pounds flux (roll scale and rich cinder). The whole is now melted down in twenty-five to thirty minutes. [In puddling gray iron, the flux is added after melting down the pig in order to cool the charge, gray iron requiring a very high temperature for melting down, and this being still further increased by the oxidation of Si.] During the melting, the puddler raises from the bottom to the surface by means of a bar parts of the pig iron charged but not completely melted, to insure the thorough melting of the whole; he then introduces a rabble (an iron rod about 10 feet long bent 90° at one end to a hook) and stirs the metal for about ten minutes so as to oxidize the Si ("clearing stage"). The temperature is now lowered and a smoky flame produced by diminishing the draught when the metal thickens somewhat. Stirring is continued to incorporate the flux and detached portions of the fix into the metal. C is oxidized to CO, which burns with a blue flame to CO_2. As the bubbles rise to the surface, they cause the metal to boil and to increase in volume, when some of the slag formed overflows through the working door into a buggy placed before it. The "boiling" lasts about fifteen minutes; it subsides and the metal and slag settle; points of white-hot wrought iron are seen to project everywhere; the iron has "come to nature." The temperature is now again raised, the wrought iron broken up into pieces by inserting a crowbar and "balled" into a large mass with the cinder hook. It is now cut, usually into six pieces, each weighing about 80 pounds, which are taken out by suspended tongs (suspended from a trolley) onto a two-wheel buggy, brought to the

"squeezer," and the resulting blooms go to the "muck-bar rolls," etc. The slag is tapped and the furnace cooled with water,[1] fettled, the bottom and the fire-bridge patched if necessary, one or two shovelfuls of rolling scale and some scrap iron from the squeezer are charged, and the next regular charge given. The length of a heat is from one and three-fourths to two hours. The bottom has to be patched once a week and renewed every three months; the furnace lasts about one year.

(5) *Chemical Reactions.* The changes that iron undergoes and the analyses of the resulting slags are shown in the subjoined tables, and graphically in Figures 170–171.[2] They refer to a gray pig iron rich in Si and Mn, puddled in Upper Silesia for wrought iron to be drawn into wire, the metal being cooled by adding cinder after melting down. In the *melting-down stage* (10.43–11.26 A.M.) Fe and Mn are oxidized as well as P and S (omitted from diagram), while C, remaining intact, increases on account of the oxidation of Fe, Mn, and P. The oxidation is caused mainly by the O of the cinder, in which the Fe_2O_3 drops from 22.31 to 9.81 per cent. In the *clearing stage* (11.28–11.35 A.M.) mainly Si is oxidized and the iron becomes white. The FeO in the slag decreases from 59.06 to 52.43 per cent., and the Fe_2O_3 from 9.81 to 6.94 per cent., showing that the oxidation is caused principally by the O of the iron;[3] the SiO_2 increases at the same time from 20.53 to 23.18 per cent. The *boiling stage* begins at 11.38 A.M.; Mn diminishes quickly and C slowly until 11.45 A.M. They then both diminish uniformly. The boiling stage reaches its maximum at 11.56 A.M.; at 12.07 P.M. the iron *comes to nature;* at 12.14 P.M. the excess of slag is tapped; the percentage of Fe_2O_3 has steadily increased. At 12.32 P.M. *balling* is begun. P is seen to decrease pretty steadily during the whole operation, about 80 per cent. being removed. A more recent set of analyses has been published by Cubillo (see references); see also E. Heyn, "Die Metallographie im Dienste der Hüttenkunde," Freiberg, 1903, p. 41.

IRON.

	Stage.								
	Melting down.		Clearing.	Boiling.		Coming to nature.		Balling.	Muck bar.
Time	A.M. 10.43	11.26	11.35	11.38	11.45	P.M. 12.14	12.19	12.32	12.42
No. of sample .	I.	II.	III.	IV.	V.	VI.	VII.	VIII.	IX.
C	2.57	2.80	2.89	2.78	2.63	1.65	1.38	0.86	0.63
Si	3.21	1.09	0.23	0.21	0.22	0.232	0.211	0.108	0.092
Mn	5.14	2.68	2.53	2.20	1.31	0.65	0.326	0.280	0.156
S	0.040	0.016	0.013	0.012	0.012	0.012	0.011	0.010	0.010
P	0.975	0.582	0.561	0.300	0.250	0.210	0.181	0.170	0.123

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[1] Danger of Explosions: "Stahl und Eisen," 1890, pp. 205, 307, 444, 535; 1892, p. 1001.

[2] Kollmann, "Zeitschrift d. Vereins deutscher Ingenieure," 1874, p. 326.

[3] Snelus, "Journal Iron and Steel Institute," 1872, p. 250.

SLAG.

Time	A.M. 10.43	11.17	11.27	11.35	11.38	11.46	P.M. 12.14	12.20	12.38	12.42
No. of sample	I.	I*a*.	II.	III.	IV.	V.	VI.	VII.	VIII*a*.	IX.
SiO_2	15.32	17.13	20.50	23.18	20.37	19.95	21.91	19.45	16.29	17.39
FeO	52.18	59.06	54.61	52.43	57.06	51.68	46.76	48.04	51.62	51.32
Fe_2O_3	22.31	9.81	7.72	6.94	9.04	11.45	12.36	13.48	19.32	17.54
MnO	6.56	9.35	12.38	12.51	10.10	11.69	15.87	14.40	8.46	9.34
P_2O_5	2.30	3.40	4.30	4.22	3.49	4.26	3.10	4.17	3.78	3.93

In order to assist in the decarburization of the iron and the removal of impurities (P, S) many fluxes ("physic") have been suggested, such as MnO_x, niter, chlorides, fluorides, alkalies; but they have, as a rule, fallen into disuse, as with metal low in P they are of no value, and when high in P the metal is better treated in the basic open-hearth furnace for low-carbon steel. In the manufacture of very low-phosphorus muck bar ($P = 0.070$ per cent.) to be used in making crucible steel, or very low-phosphorus acid open-hearth steel, fluxes are still used.

(6) *Results.* A charge of 500 pounds pig gives about 510 pounds muck bar, the increase being due mainly to reduction of iron from the fettling. The waste, about 12 per cent., decreases as the non-metallic elements, Si and P, increase;[1] it is increased also by a deficiency of cinder. There is usually required 1 ton of coal per ton of muck bar. Two men form a crew, working in eight- to ten-hour shifts. The following table by C. D. Wright[2] gives the cost of 1 gross ton muck bar in 1890 in the United States. In a general way it may be said that the cost of muck bar is 75–90 per cent. larger than the price of the pig.

No. of puddling furnace.	MATERIALS.						Labor.	Officials and clerks.	Fuel.	Supplies and repairs.	Taxes.	Total.
	Pig iron.	Scrap.	Ore.	Gross total.	Value of cinder scrap, etc.	Net.						
47	$10.993	Reckoned with pig	$2.730	$13.723	$0.170	$13.553	$7.746	$0.244	$2.028	$0.313	$0.030	$23.914
33	16.305	0.366	1.426	18.097	0.362	17.735	7.155	0.324	0.711	0.870	0.053	26.848
20	5.903	10.371	0.386	16.660	1.041	15.619	6.969	0.810	1.906	0.758	0.081	26.143
12	13.046	3.757	1.349	18.152	0.845	17.307	7.405	0.521	1.838	0.957	0.052	28.080
6	15.680	. . .	1.391	17.071	0.365	16.706	7.380	0.623	2.219	1.104	0.086	28.118

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[1] Scarf, "Journal Iron and Steel Institute," 1891, I, p. 150.

[2] Sixth Annual Report Commissioner of Labor, 1890, pp. 110 and 114.

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Calorific efficiency of the puddling furnace (see Cubillo, "Journal Iron and Steel Institute," 1892, I, p. 245).

(7) *Mechanical Treatment of Puddled Ball.*[1] The puddled ball of wrought iron when withdrawn from the furnace is mixed up with fluid cinder, which has to be expelled before it can be rolled into muck bar. In this country

(*a*) Squeezers, invented by Ralston in 1840, are used almost exclusively, while in Europe the steam hammers are prevalent. A squeezer (Figure 172)[2] consists of a ribbed roller, *b*, 4–4½ feet in diameter, 22–27 inches high, revolving in the direction of the arrow at the rate of five revolutions per minute in the stationary ribbed eccentric casing *a*. It is driven from below by gearing, the pinion shaft being held by the housing posts, *E*, and by cross arms. The ball is dropped at *C* onto the annular plate of the roller, is compressed while passing through the narrowing open space, and delivered, in ten to fifteen seconds, at *d*; the space at *c* is about 20 inches wide, that at *d* 5–8 inches. Water is sprayed onto the squeezer to keep it cool; coming in contact with the wrought iron and slag, explosions are of frequent occurrence. The roller lasts one year, the casing two years.

In comparison with steam hammers, squeezers have the disadvantage that the puddled ball must always be of the same size, and that even then the slag is not completely expelled; their advantage lies in the fact that they require little labor, and therefore work cheaply.

(*b*) Muck Rolls.[3] The puddle ball (2 feet long and 7 inches in diameter), having been compressed in the squeezer, is taken straight to the muck rolls to be rolled into muck bar. Figure 173 represents a train of two-high rolls, through which the puddle bar is passed to and fro, and rolled into muck bar and then allowed to cool. Muck bar is about ⅜ inch thick and 2, 4, and 6 inches wide. A 2-inch bar takes eight to nine passes, a 4-inch seven to eight passes, a 6-inch six to seven passes. The muck bar is now cropped, *i.e.*, cut up by powerful crocodile shears into short pieces of equal length, say 24 inches.[4] A number of these are piled (laid flat on top of onc another), forming a packet, tied together by a wire, and brought in the

(*c*) Reheating Furnace[5] to a welding heat. The Stubblebine Eureka reheating furnace in use at the Bethlehem Steel Company's Works, South Bethlehem, Pennsylvania, for reheating muck-bar packets, is shown in Figures 174–175. It is a reverberatory furnace, on the working bottom, *e*, of which the packets of muck bar are heated. The flame from the coal, burned on grate *a*, passes over the fire-bridge *b* onto the hearth *d*, at the end of which are three flues, the main flue, *f*, leading into the stack *h*, and the two side flues, *f'*, leading into the heating chambers. Air from a fan entering the furnace at *i* is split into two branches leading to the heating chambers, and issuing forth in the mixing flues, *j*. The superheated air burns any products of imperfect combustion and enters the hearth on either side of the fire-bridge at *k* to assist in completely burning the combustible matter of the flame. In a reheating furnace 8 x 4 feet about 1 ton of

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[1] Plant of Empire Rolling Mill Company, Cleveland, Ohio: "Iron Trade Review," January 31, 1901.

[2] Wedding, "Das Schmiedbare Eisen," p. 754; Frésore, "Revue Universelle des Mines," 1885, XVIII, p. 195; Catalogue of Lewis Foundry and Machine Company, Pittsburgh, Pennsylvania. Squeezer of Lloyd, Booth and Company, Youngstown, Ohio: "Iron Age," March 20, 1890.

[3] Catalogue Lewis Foundry and Machine Company, Pittsburgh, Pennsylvania.

[4] See Gubbin, "Iron Age," October 24, 1890.

[5] "Stahl und Eisen," 1891, p. 558; 1903, p. 1378; "Iron Age," January 28, 1904; *op. cit.*, September 12, 1901; January 6, 1902.

muck bar will form a charge, and twelve charges will be reheated in twenty-four hours. The oxidized iron on the surface will peel off and combine with the brick bottom (covered for protection with iron ore) to a slag (heating cinder) with SiO_2, 20–30 per cent.; Fe, 50 per cent., usually in the form of FeO. A bottom lasts about six months. Reheating furnaces with hearths 7 x 16 feet and grates 7 x 3½ feet are common. From the reheating furnace the packets are taken with tongs suspended from an overhead inclined rail to the

(*d*) Finishing Rolls, which are of similar general construction as the muck rolls. A finished bar weighs usually 10 per cent. less than the packet charged into the reheating furnace; it is cut into the required lengths and is ready for the market.

(8) *Direct Puddling of Iron,* in which the liquid metal is transferred from the blast furnace to the puddling furnace, thus avoiding the melting down of pig in the puddling furnace, has been repeatedly suggested, but has not found much favor on account of the chemical reactions that take place in the melting down of the pig, which cannot be had with direct metal. Bonehill[1] is working a direct-metal plant successfully at Hourpes (Belgium). The blast-furnace metal is tapped into a stationary receiver (holding 35 tons metal, 23–27 inches deep) heated with waste gas from the blast furnace. From this the metal is tapped into a ladle placed on scales and then a weighed amount poured into the puddling furnace. This is heated with gas from four producers; it has Siemens regenerators, and converts 2,200–2,640 pounds direct metal in one-half hour into a bloom; the time between charges is less than one hour. Besides the general saving in doing away with casting of pig at the blast furnace and returning scrap, there is a saving of labor (eight puddlers in twenty-four hours treat 12 metric tons, or 1½ tons = 3,300 pounds per man *versus* 2,035 pounds in the ordinary system); in fuel (20 per cent. + 2 per cent. [keeping metal fluid in receiver] = 22 per cent. *versus* 100 + per cent.); in iron (loss 6 *versus* 16 per cent.); in life of furnace (long *versus* short); and in quality of product, which is more uniform and stronger than the ordinary product.

	C	Si	P	S
Mottled iron: —				
When poured into the receiver	2.95	0.43	1.40	0.50
When tapped after four hours from receiver . .	2.85	0.40	1.35	0.40
Puddled bar	. . .	0.23	0.45	0.22
Forge pig iron: —				
When poured into the receiver	3.30	0.65	1.15	0.32
When tapped after four hours from receiver . .	3.15	0.60	1.10	0.25
Puddled bar	. . .	0.28	0.40	0.20
Wrought iron from puddled bar	. . .	0.15	0.20	0.04

See also Puddling Blast-furnace Metal at Milwaukee: "Iron Age," February 23, 1888.

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[1] "Journal Iron and Steel Institute," 1895, II, p. 43; "Revue Universelle des Mines," 1896, XXXIII, p. 22.

(9) *Other Furnaces.* Besides the ordinary puddling furnace discussed, furnaces with preheating chambers, double furnaces[1] with charges up to 1,300 pounds, heated with gaseous fuel and having regenerative chambers, are used in some European works. Examples are: the Springer furnace,[2] the Pietzka furnace,[3] and others.[4]

(10) *Mechanical Puddling.* As puddling requires a great deal of skill and is very hard on the men, attempts have been made to replace hand labor in stationary furnaces by mechanical devices,[5] but none of them have been successful, as the cost of installation and of repairs was too great, and then the work not altogether satisfactory. Furnaces with revolving hearths were used for a number of years, but they also have been given up. The leading furnace was that of Danks,[6] which was used in Pittsburgh for ordinary puddling work and in Cleveland for making wrought iron to be used in the open-hearth furnace. It is a short, cylindrical, horizontal furnace, revolving on two pairs of friction rollers at the ends, and driven by a circular gear around the middle. The fettling is sintered down in layers, and liquid metal is charged from a ladle. A charge of 1½ tons metal was puddled in fifteen minutes. The puddled ball was removed by a long-handled paddle suspended from a crane, carried to a hammer, compacted, and cut into blocks for the open-hearth furnace. A new mechanical puddling machine by Roe[7] is said to be doing satisfactory work at Pottstown, Pennsylvania.

(11) *Comparison of Methods.* Bell[8] gives the following table comparing the different methods of puddling:—

	C	Si	S	P
Cleveland pig contains, say	3.50	1.75	0.15	1.55
Cleveland pig when refined	2.50	0.091	0.11	0.50–0.75
Cleveland pig refined and dry puddled . . .	. . .	0.19	0.01	0.26
Cleveland pig hand puddled, wet	. . .	0.159	0.041	0.324
Cleveland pig mechanically puddled	. . .	0.109	0.032	0.10–0.17

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[1] Loomis-Pettibone Furnace: "Iron Age," February 11, 1892.

[2] "Stahl und Eisen," 1889, pp. 554, 776; "Journal Iron and Steel Institute," 1889, II, p. 424; 1890, II, p. 529.

[3] "Stahl und Eisen," 1889, XV; "Journal Iron and Steel Institute," 1890, II, p. 527.

[4] "Iron Age," February 21, 1889; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1903, p. 441.

[5] Howson, "Journal Iron and Steel Institute," 1877, p. 416; Jeans, *op. cit.*, 1882, I, p. 143; Dingler, "Polytechnisches Journal," CLXXXV, p. 242.

[6] "Journal Iron and Steel Institute," 1871, I, p. 246; II, p. 258; 1872, I, pp. 278, 293, 311; "Transactions American Institute of Mining Engineers," II, p. 28; Greenwood, "Steel and Iron," p. 289.

[7] "Transactions American Institute of Mining Engineers," 1903, XXXIII, pp. 551, 1041.

[8] "Principles," p. 388.

[illegible]

[illegible]

[illegible]

CHAPTER V.

PRODUCTION OF STEEL.

SECTION 47. **Literature**: manuals quoted on page 1, further:—

Howe, H. M., "The Metallurgy of Steel," Volume I. New York, 1890.

Campredon, L., "L'Acier." Paris, 1890.

Campbell, H. H., "Manufacture and Properties of Iron and Steel." New York, 1903.

Harbord, F. W., and Hall, J. W., "The Metallurgy of Steel." London. Philadelphia, 1904.

SECTION 48. **Introductory.** Steels may be grouped under the two heads: Carbon (regular) Steels and Special (alloy) Steels. Carbon steels are essentially alloys of iron and carbon; special steels are alloys of iron and carbon with other elements, such as tungsten, chrome, manganese, nickel, etc., which give them peculiar properties.

The properties of carbon steels, or regular steels, are determined not only by their chemical composition and heat treatment, but also by the process of manufacture.

Steel is produced by the Bessemer, the Open-hearth, the Cementation, and the Crucible Processes, furnishing Bessemer (acid and basic) steel, Open-hearth (acid and basic) steel, Cement steel, and Crucible steel. The relative importance of the processes in this country is shown by the following table:[1]—

	Acid.		Basic.		Total.
	Gross tons.	Per cent.	Gross tons.	Per cent.	Gross tons.
Bessemer	9,306,471	61.6	. . .	. . .	9,306,471
Open hearth	1,191,196	7.9	4,496,533	29.7	5,687,720
Crucible	125,000	0.8	. . .	. . .	125,000
Totals, 1902	10,622,667	70.3	4,496,533	29.7	15,119,200
Totals, 1901	9,854,602	73.1	3,618,993	26.9	13,473,595

Bessemer steel is about $5 to $6 higher in price than the pig iron from which it is made; Open-hearth steel about $7 to $8 higher; Crucible steel is about 35 cents a pound, the price fluctuating very little with the cost of the raw materials.

SECTION 49. **The Bessemer Process in General.** In this process finely divided air is forced through melted and superheated pig iron for the purpose of oxidizing Si, Mn, C, and in the basic process also P (and S), thus converting the pig iron into steel. The heat required to

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[1] "Mineral Industry," 1902, XI, p. 364.

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AND
NS.

BESSEMER WORKS, South Chicago.

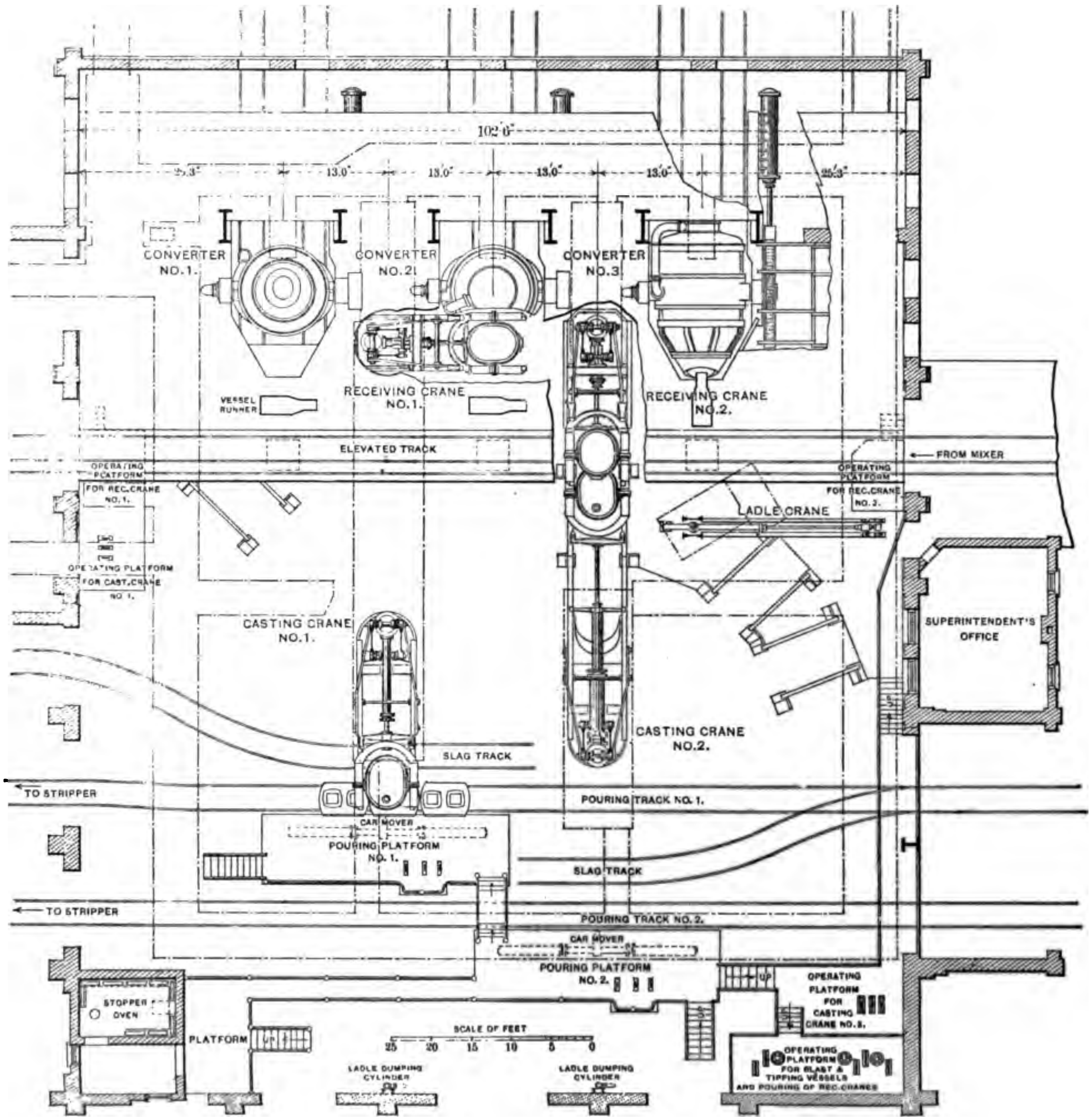

Fig. 176. Plan.

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BESSEMER WORKS, South Chicago.

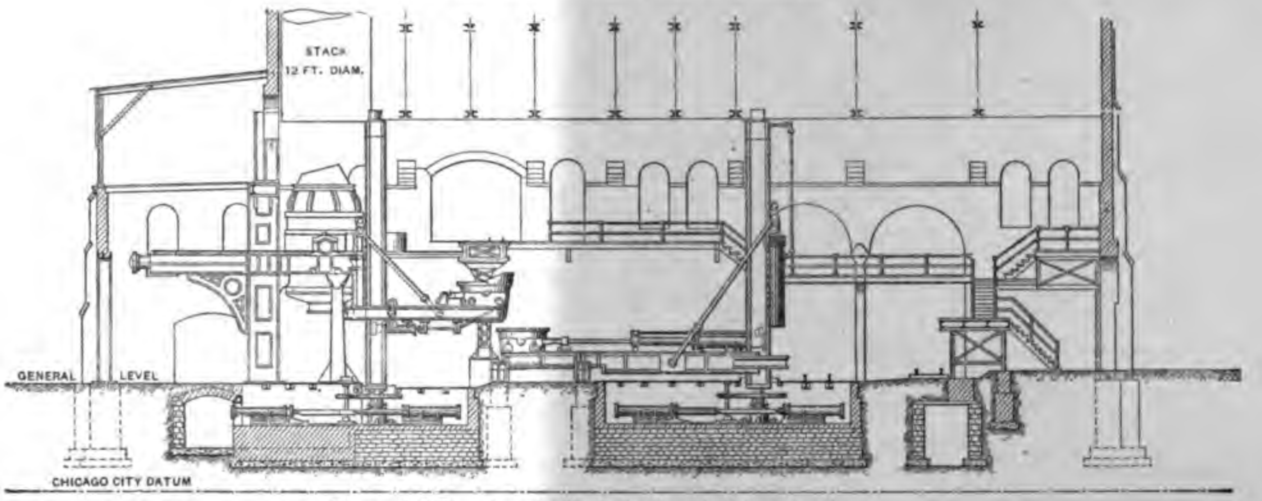

Fig. 177. Longitudinal Section, looking South.

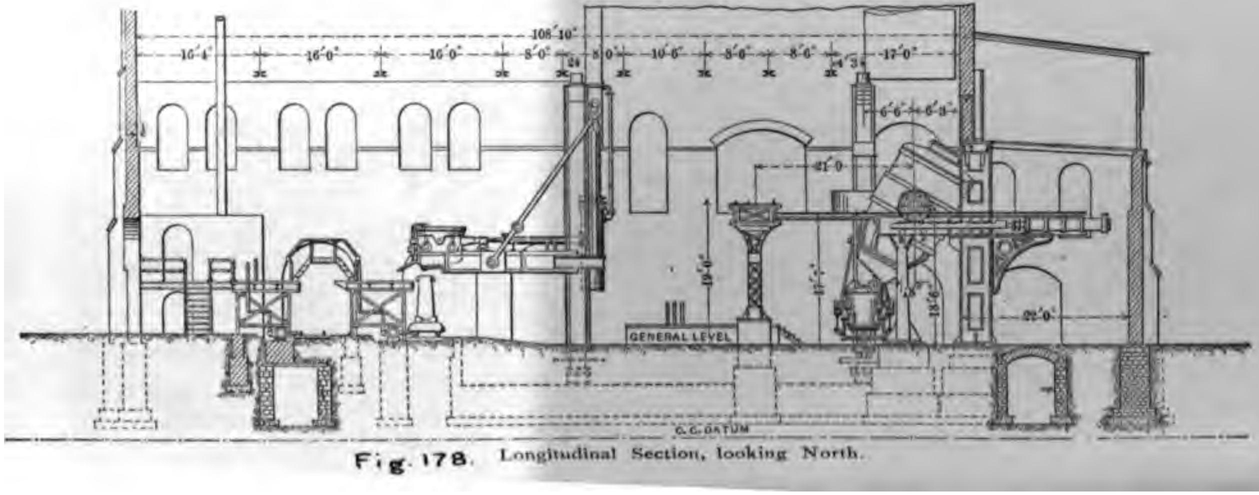

Fig. 178. Longitudinal Section, looking North.

keep the steel and slag in a fluid state is furnished principally by the burning of Si in the acid and P in the basic process.

$$Si + O_2 = SiO_2 + 219{,}240 \text{ cal. } (1 \text{ kg. } Si + 7{,}830 \text{ cal.})$$
$$P_2 + O_5 = P_2O_5 + 357{,}120 \text{ cal. } (1 \text{ kg. } P + 5{,}760 \text{ cal.})$$

A blowing can either be carried just far enough to oxidize the desired amount of C, or, and this is the common practice, the metal is overblown to eliminate nearly all the C, and the low-carbon iron formed is then recarburized by the addition of spiegeleisen, ferromanganese, or pulverized coke (Darby process, 1889).

The process was invented[1] in 1855 by Henry (the late Sir Henry) Bessemer. The product of a blow, a spongy, burnt iron, became homogeneous and sound only through the addition of spiegeleisen (Mushet's patent of 1856). This not only recarburized the iron, but removed the O, the Mn combining with it and entering the slag. The many difficulties encountered at first were overcome only in 1858,[2] Goransson having much to do with it. In 1861 Bessemer first used ferromanganese instead of spiegeleisen, and thus laid the basis for the production of low-carbon steel. The process revolutionized the iron industry. Bessemer thought at first that good steel could be made from any kind of pig iron, but he soon found that S and P were not eliminated, and that he therefore could use only low-S and low-P pig iron. This difficulty was removed in 1878 by Thomas and Gilchrist, who replaced the acid lining of the Bessemer converter by a basic lining, made of burnt dolomite, and added lime to the charge.

Two processes are distinguished at present — the *Acid Bessemer Process* and the *Basic Bessemer Process*. The basic Bessemer process, which is more expensive than the acid, has become prominent, especially in Germany, where there is a great abundance of ore suited for it, while acid Bessemer ores are scarce.

A. THE ACID BESSEMER PROCESS.

SECTION 50. **General Arrangement of Plant.**[3] The good arrangement of a Bessemer plant is of tremendous importance on account of the great quantities of material that have to be handled in large units. The converter charges are large, and follow each other in rapid succession. A modern plant has a very different aspect from one built ten years ago. This is well illustrated by comparing the plant of the South Chicago Works built by R. Forsyth in 1880,[4] with that finished February, 1900, and represented in plan and two longitudinal sections by Figures 176–178.[5] This plant has three eccentric 14-ton converters (vessels) placed in a straight line. A converter (*e.g.*, Number 3) receives its weighed charge from a metal ladle which is brought in front of it (dotted square, Figure 176) on an elevated track from the mixer by a locomotive. The ladle is emptied into the vessel runner discharging into converter. The trough is swung aside and the converter brought into a vertical position (Figure 177) and the

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[1] "Transactions American Institute of Mining Engineers," XIX, p. 810, Kitson; XXVI, p. 980, Weeks; XXVIII, p. 745, Phillips; "Journal Iron and Steel Institute," 1886, II, p. 638, Bessemer; 1896, I, p. 470, Summary.

[2] Åkerman, "Transactions American Institute of Mining Engineers," XXII, p. 265.

[3] "Stahl und Eisen," 1900, p. 357; 1901, pp. 1097, 1168, American Bessemer Plants; *op. cit.*, 1901, p. 1102, Sparrow's Point; "Iron Age," October 11, 1900, Youngstown, Ohio; *op. cit.*, June 7, 1904, Lackawanna Steel Company, Buffalo, New York; "Journal Iron and Steel Institute," 1901, I, p. 299, Barrow, England; Howe, "Steel," p. 316.

[4] "Transactions American Institute of Mining Engineers," XII, p. 272.

[5] "Iron Age," July 6, 1899.

charge blown, the metal being 12–20 inches deep. In the meantime the pig-metal ladle is pulled out and replaced by a spiegel ladle. The converter when blown is turned down and receives its spiegel for recarburizing; the heat of steel is then discharged (Figure 178) into the tilting ladle of the receiving crane Number 2. (Receiving crane Number 1 is shown in that position before converter Number 2.) The steel is emptied into the casting ladle of casting crane Number 2, which is swung around, and then the steel teemed into the ingot molds on pouring track Number 2 (as shown with casting crane Number 1). The mold trucks are pushed along by the car mover, and when a train of them has been filled it is taken by a locomotive to the stripper, which removes the molds from the ingots. The three converters are served by two receiving cranes, and each of these by a casting crane, which fills the molds arriving at the right on pouring tracks Numbers 1 and 2 respectively, and leaving at the left. Casting crane Number 2, however, reaches over the casting track of Number 1. The cranes are operated by hydraulic power, and the ladles on the arms of the cranes are moved back and forth by horizontal plungers. The converters, the blast, and the tapping of the vessels of the receiving cranes are managed from the platform (Figure 176) in the lower right-hand corner; in front of it, and a few steps lower down, is the operating platform for casting crane Number 2, while the pouring platform is to the left of it and a few steps higher up. The platform for operating receiving crane Number 2 is to the right near the elevated metal track. On the left side of the building are the platforms for operating receiving crane Number 1 and casting crane Number 1. On the right side is a ladle crane for exchanging ladles. The track on which cars carrying the bottoms of converters run is not given. All the slag made is loaded into an iron car on the slag track. This double pouring is said to facilitate the chemical reactions after recarburizing and to make the steel more uniform.

A cross section through the Converting Department of the Edgar Thomson Works, Homestead, near Pittsburgh, Pennsylvania, is given in Figure 179.[1] The building is 160 x 80 feet and has four 20-ton concentric converters in a straight line, a pair of converters being served by one crane. The section shows to the left the metal ladle *C*, the vessel runner *D*, the converter *A* in a horizontal position receiving its spiegeleisen from the ladle *f*, the crane *j* with ladle *j'*, the ingot mold *g* on pouring track *p*, the operating platform *t* and the crane *u* serving to exchange the ladles which arrive and leave on the track *v*. The converter, which can be swung through an arc of over 180°, is turned to a horizontal position with the nose pointing to the vessel runner *d*; it receives its pig iron from the metal ladle *c*, brought there by a switch engine from the mixer; is turned up, blown; turned down to the position shown in the figure and receives its spiegeleisen; the heat is then poured into ladle *j* and the steel teemed into ingot molds, *g*.

SECTION 51. **Raw Materials.** These are Bessemer pig, spiegeleisen, and ferromanganese.

(1) *Bessemer Pig* (analyses, see p. 99). The molten iron for conversion is obtained either by remelting the pig iron in the cupola (cupola metal), or by taking it from a mixer which receives the molten metal direct from the blast furnace (direct metal).

(*a*) Cupola Metal.[2] The cupola must be so located as to permit readily conveying the metal to the converter and removing the slag and *débris* to the dump. A pair of cupolas

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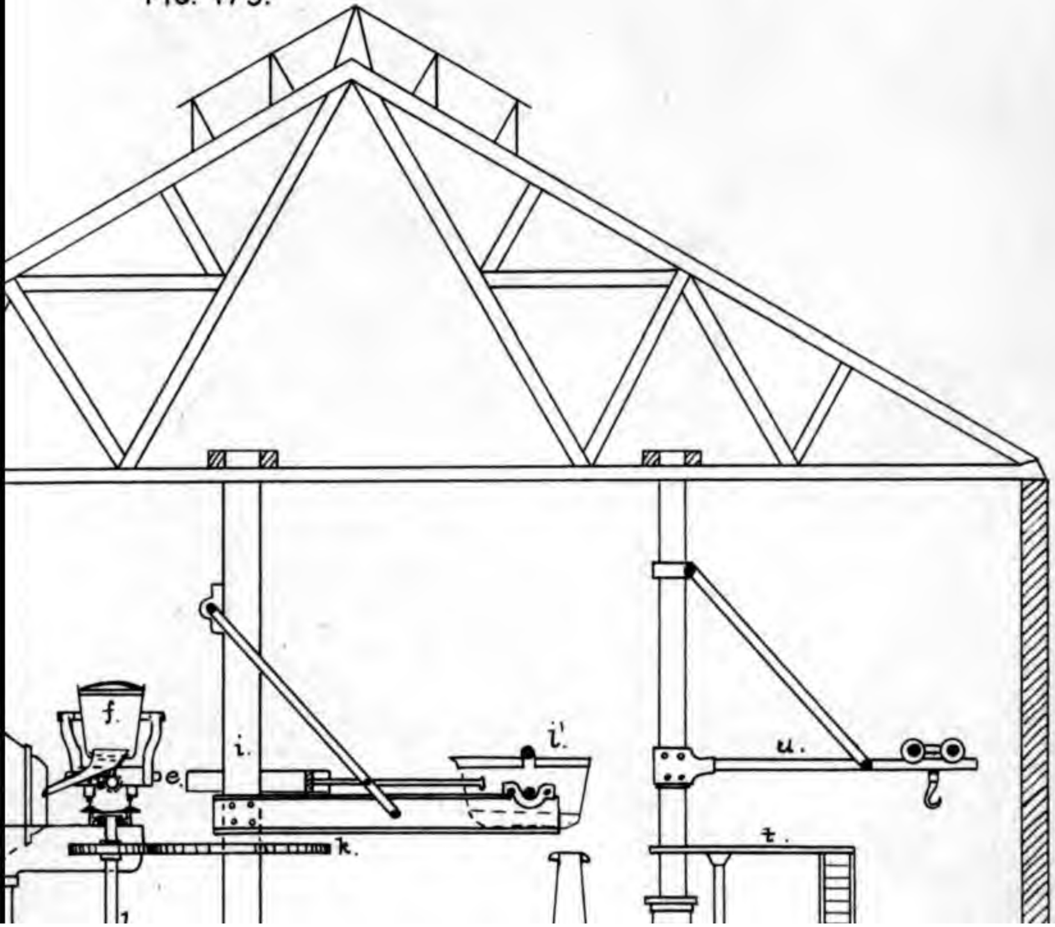
[1] "Stahl und Eisen," 1894, Plate II.

[2] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 170; Howe, "Transactions American Institute of Mining Engineers," XIX, p. 1150; Rogers, "Journal American Foundrymen's Association," 1900, XI ^{1}, p. 402.

FIG. 179.



charge blown, the metal being 12–20 inches deep. In the meantime the pig-metal ladle is pulled out and replaced by a spiegel ladle. The converter when blown is turned down and receives its spiegel for recarburizing; the heat of steel is then discharged (Figure 178) into the tilting ladle of the receiving crane Number 2. (Receiving crane Number 1 is shown in that position before converter Number 2.) The steel is emptied into the casting ladle of casting crane Number 2, which is swung around, and then the steel teemed into the ingot molds on pouring track Number 2 (as shown with casting crane Number 1). The mold trucks are pushed along by the car mover, and when a train of them has been filled it is taken by a locomotive to the stripper, which removes the molds from the ingots. The three converters are served by two receiving cranes, and each of these by a casting crane, which fills the molds arriving at the right on pouring tracks Numbers 1 and 2 respectively, and leaving at the left. Casting crane Number 2, however, reaches over the casting track of Number 1. The cranes are operated by hydraulic power, and the ladles on the arms of the cranes are moved back and forth by horizontal plungers. The converters, the blast, and the tapping of the vessels of the receiving cranes are managed from the platform (Figure 176) in the lower right-hand corner; in front of it, and a few steps lower down, is the operating platform for casting crane Number 2, while the pouring platform is to the left of it and a few steps higher up. The platform for operating receiving crane Number 2 is to the right near the elevated metal track. On the left side of the building are the platforms for operating receiving crane Number 1 and casting crane Number 1. On the right side is a ladle crane for exchanging ladles. The track on which are carrying the bottoms of converters run is not given. All the slag made is loaded into an iron car on the slag track. This double pouring is said to facilitate the chemical reactions after recarburizing and to make the steel more uniform.

A cross section through the Converting Department of the Edgar Thomson Works, Homestead, near Pittsburgh, Pennsylvania, is given in Figure 179.[1] The building is 160 x 85 feet and has four 20-ton concentric converters in a straight line, a pair of converters being served by one crane. The section shows to the left the metal ladle *C*, the vessel runner *d*, the converter *A* in a horizontal position receiving its spiegeleisen from the ladle *f*, the crane [illegible] ladle *f′*, the ingot mold *g* on pouring track *p*, the operating platform *l* and the crane [illegible] to exchange the ladles which arrive and leave on the track *s*. The converter, which [illegible] swung through an arc of over 180°, is turned to a horizontal position with the nose [illegible] the vessel runner *d*; it receives its pig iron from the metal ladle *c*, brought there [illegible] engine from the mixer; is turned up, blown; turned down to the position shown, and receives its spiegeleisen; the heat is then poured into ladle *f* and the steel [illegible] molds, *g*.

SECTION 51. **Raw Materials.** These are Bessemer pig, spiegeleisen [illegible]

(1) *Bessemer Pig* (analyses, see p. 99). The molten iron for [illegible] either by remelting the pig iron in the cupola (cupola metal), or by taking [illegible] receives the molt[illegible] (direct metal). [illegible]

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in front elevation and vertical section is shown in Figures 180 and 181. The cupola differs slightly from that used in remelting cast iron for foundry purposes. It is 8–10 feet in outside diameter, 20–26 feet high, often has a slight bosh; the lining is of fire brick; above the bosh it is 12–18 inches thick, at the bosh and tuyères 24 inches; the lower part, of sandstone, is 24 inches thick. Formerly it had about eight tuyères; at present it has a wind box from which sixty to one hundred 2- to 3-inch openings in two to three rows pass into the furnace; the tuyère area should be at least two and one-half times that of the air box. About 30,000 cubic feet air are necessary to melt 1 ton of pig iron. The blast pressure is 12–18 ounces. The pig-iron mixture is made up to run about Si, 1.75 per cent.; Mn, 0.75 per cent.; S, below 0.05 per cent.; P, below 0.10 per cent.; Cu, below 0.10 per cent. The Si is high, as about 0.5 per cent. is burned in remelting, the melted pig running usually 1–1.5 per cent. Si. The amount of Si burned is larger with a low than a high temperature of the cupola. Sometimes scrap steel is charged with the pig, provided the Si in the pig is high enough to stand this dilution. The coke required is 7–10 per cent. on the pig; with 6 per cent. there is danger of chilling. The coke must be low in S, under 0.10 per cent., as this is absorbed by the pig. The addition of manganese ore to the charge aids in carrying the S into the slag.[1] The limestone required to flux the coke ash and the sand on the pig is about 3.5 per cent. From 0.3 to 0.5 per cent. Si and some manganese are slagged in melting. Campbell[2] gives the following table for loss in iron in 7-foot cupolas: —

Pig iron charged, pounds	835,600	Fe in slag, per cent.	8.77
Coke, pounds	75,750	Fe in slag, pounds	3,529
Limestone, pounds	15,250	Fe in slag, per cent. of pig iron charged	0.42
Cupola slag, pounds	40,200		

In starting a newly lined cupola, it is fired with wood, filled two-thirds full of coke, and then charged. The pig given at first must run high in Si, say 2.5 per cent., as both cupola and converter are comparatively cold; the burning of this extra amount of Si will bring the metal in the converter and the converter to the required high temperature. The melting takes about two hours. A cupola with 6 feet inner diameter will melt in twenty-four hours 200–250 tons of pig; the lining lasts not over a week, as a rule not much over three days. The required amount of metal is tapped from the cupola into a ladle standing on scales, and the metal hauled to the mouth of the converter by a locomotive, rope-haulage, an overhead traveling crane, electric motor, etc.

(*b*) Direct Metal. The composition of the metal as it comes from large coke blast furnaces varies too much in its Si (and S) content to permit its being poured directly from the ladles into the converter. It is therefore collected in a large reservoir, called mixer,[3] and poured from this into a ladle standing on scales, and goes then to the converter. Direct metal is used only (excepting Swedish practice) in large plants having more than one blast furnace and in which the distance between blast furnace and converter plant is not more than two miles.

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[1] "Stahl und Eisen," 1902, p. 415; 1903, p. 1135; "Iron Age," November 19, 1903, p. 14.

[2] "Manufacture and Properties of Iron and Steel," 1903, p. 170.

[3] "Stahl und Eisen," 1890, p. 26; 1901, p. 1099; 1902, p. 307; "Iron Trade Review," April 17, May 15, 1902; October 15, 1903; April 28, 1904; "Iron and Coal Trades Review," LV, p. 154; "Revue Universelle des Mines," 1894, XXIV, p. 291; "Journal Iron and Steel Institute," 1895, I, p. 67; "Colliery Guardian," November 25, December 16, 1898; "Cassier's Magazine," 1900, XVII, p. 267; Harbord, *op. cit.*, p. 16.

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[illegible]

A sample of each ladleful of metal (say from a train of six ladles) before going to the mixer is taken and the Si (and S) determined. The average Si (and S) is calculated and the result communicated to the blower, who guides the operation of the converter accordingly. Variations in the Si of irons charged into and drawn from the mixer of the Cockerill Works, Séraing, Belgium,[1] are shown in Figure 182, in which the heavy lines represent the blast-furnace metal, the dotted lines the mixer metal. A similar set of figures is given by Harbord, "Metallurgy of Steel," p. 20. Fracture tests of small ingots cast in iron molds give a very close idea of the percentage of Si. Some S is eliminated in the mixer,[2] but no Si or C. In order to prevent the iron from chilling on top, it is heated by an oil jet blown with steam or air. Sticks of wood are often charged to consume the free oxygen of the air. As the temperature of the direct metal is higher than that of the cupola metal, the Si content can be lower, viz., 0.75–1.00 per cent. *versus* 1.50 per cent. In Sweden, where small charcoal furnaces, running on the same charge, furnish metal of very uniform composition, the metal goes from the blast furnace straight into the converter.[3]

Stationary Mixers, formerly used in Europe, have been abandoned. The Mixer of Hoerde, Germany, is represented in Figure 136.

The Jones Mixer, formerly used at the Edgar Thomson Works, Pittsburgh,[4] is represented in side elevation in Figure 183. There were in use two mixers, each with a capacity of 80 tons. The mixer is 17 feet long, 10 feet wide at the bottom and 11 feet at the top; the height on the filling side is 8½ feet, on the discharging side 5½ feet; the spring of arch is 1 foot; the discharge is 1 foot above the bottom, thus leaving always a bath of metal for equalizing temperatures. The mixer is emptied by tapping. The bottom plate is 1 inch and the side and roof plate ¾ inch thick. The bottom is stiffened by five 5-inch I-beams, the sides by six 8-inch I-beams, the back by three 9-inch I-beams. The brick lining is 13½ inches thick on bottom and sides and 9 inches in the roof. In the roof are the receiving funnel and two poking holes; on the discharge side one pouring spout and two tapping spouts. The mixer swings on a shaft 6 inches in diameter and is raised and lowered by a pinion engaging in a vertical rack. The variations in blast-furnace metal were 0.4–4.0 per cent. Si; the average mixer metal contained 1.5 per cent. Si. With the increase of size of blast furnaces the mixers had to be enlarged. The common form of

The Pittsburgh Mixer is shown in Figures 184–185. It is bowl-shaped, runs on rollers, is rocked by a pair of piston rods on either side, actuated by hydraulic power. On either side is a door, through which an oil jet can be introduced. The metal, poured in through the opening in the roof, is discharged from the spout in the side; receivers holding as much as 325 tons metal are in operation, the usual capacity is 125–200 tons.

The Wellman Mixer,[5] represented in Figures 186–188, is in use at the South Chicago Works of the United States Steel Company. It is a horizontal steel cylinder of 150 tons capacity, resting on six friction rollers, *D*. On the top, at one end, is the receiving spout *B*;

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[1] "Revue Universelle des Mines," 1894, Plate 8, Figure 6.

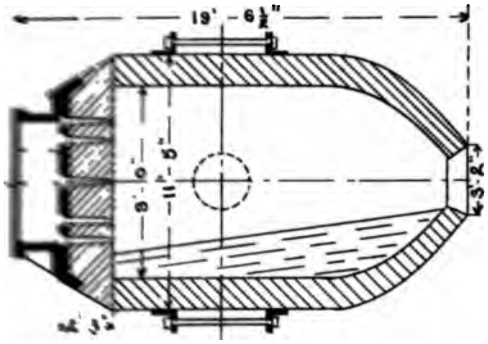
[2] See p. 104.

[3] Åkerman, "Transactions American Institute of Mining Engineers," XXII, p. 268.

[4] "Stahl und Eisen," 1890, p. 26. Patent decision: "Iron Age," LXVII, January 24, 1901, p. 10; LXIX, May 22, 1902, p. 7; "Iron Trade Review," XXXV, May 22, 1902, p. 35.

[5] A mixer, 20 feet long and 10 feet in diameter inside, when charged with 6 feet of metal holds 50 tons.

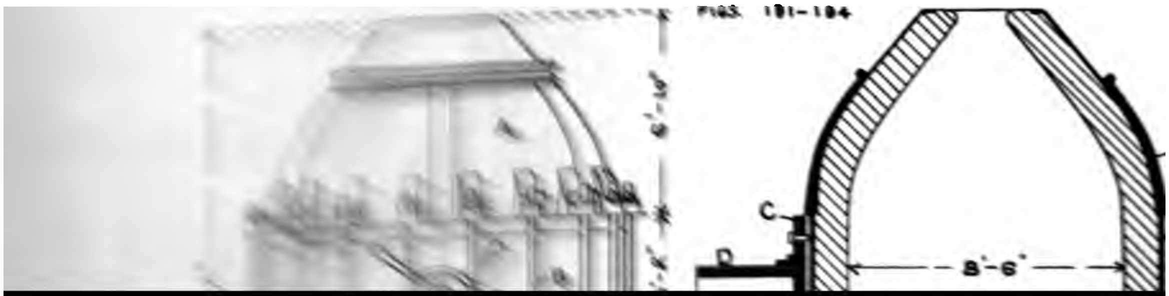
FIG. 190.



CONCENTRIC CONVERTER.
MARYLAND STEEL CO.

CONCENTRIC CONVERTER.
EDGAR THOMPSON WORKS.

FIGS. 191-194



on the side, near the opposite end, the pouring spout. The cylinder is revolved by rack and pinion *C*. It is lined with two courses of 9-inch fire brick backed by a 2-inch layer of ganister. The immersed bricks last six weeks, those above the metal two to three months. On the end of the cylinder near the discharge is an opening, through which an oil jet is introduced to keep the slag from chilling; the opening also serves to insert a bar for stirring up the metal, if it shows a tendency to layer when cold or when it is low in Si. Figure 186 shows the arrangement of the two mixers of the converter plant: $E=$ track on which hot-metal ladles arrive; $F=$ elevator which raises a ladle to pouring track *G*, and, while rising, the platform with the ladle is turned 90°. The receiving spouts, *B*, of the mixers face the pouring track, while the pouring spouts, *A*, face one another, discharging the metal into the receiving hot-metal ladle *H* on the ground floor. The pouring hot-metal ladle on track *G* is brought by rope-haulage to the mixer, emptied, and then hauled onto platform F'; this is lowered to floor, and while lowering turned again 90°, when the empty ladle is run off to the blast furnace on track E'.

(2) *Spiegeleisen* (analyses, see p. 94). This is melted down in a cupola similar to the metal cupola, only smaller in diameter, say 4 feet. There are fewer of the spiegel cupolas than there are of the metal cupolas, as a converter charge receives for rail steel about 10 per cent. spiegel. The loss in Mn in remelting is about 10 per cent.

(3) *Ferromanganese* (analyses, see p. 94) is brought to a red heat in a reverberatory furnace and shoveled into the converter, usually 0.8–1.0 per cent. on the weight of the charge.

SECTION 52. **The Converter and Blowing Engine.** Converters may be classed as fixed and rotating, as eccentric and concentric, as side- and bottom-blown. Excepting perhaps a few 1 to 3 ton baby-Bessemer converters, all converters at present are rotating and bottom-blown; some are eccentric and others concentric. Baby-Bessemer converters treat 1 to 3 tons metal in a heat, small converters 5 tons, medium size 10 tons, large size 20 tons; the capacity of a converter is 50–65 cubic feet per ton of iron to be blown.[1]

An eccentric rotating converter is shown in Figures 177–178 while blowing and pouring. Figure 189[2] represents it in a horizontal position when receiving metal or just before pouring. The metal is received through the nose into the belly and poured from the belly. In the concentric 18-ton converter of the Maryland Steel Company (Figure 190),[3] the cast iron is received on one side (iron side) and the steel poured on the opposite side (steel side), as was shown in Figure 179. The details of the concentric converter of the Edgar Thomson Works are shown in Figures 191–194.[4] It is a 20-ton converter; usually 16–17 tons make up a charge. The main shell *A* is attached by key bolts *B* to the trunnion ring *C* with trunnions *O* and *O'*. To the lower part of the main shell is fastened, by means of key bolts *E*, the shell *F* of the bottom.

In the early converters, bottom and sides were permanently joined, which caused long delays in repairing the tuyères, when these had been eaten out by the slag. Holley and Pearse in 1869 detached the bottom from the vessel, thus allowing a converter bottom to be replaced quickly by a new one heated so as to be ready for immediate use.

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[1] Howe, "Journal Iron and Steel Institute," 1890, II, p. 129.

[2] Howe, "Steel," p. 341.

[3] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 156.

[4] "Stahl und Eisen," 1894, p. 251.

To the shell is attached the tuyère plate G, through which pass the tuyères H. Plate G forms one end of the tuyère box I, and the lid J, joined to the sides by key bolts, the other. Compressed air enters the hollow trunnion D' through the rotary slip joint K, passes through the goose neck M and the tuyère pipe N into the tuyère box. The trunnion D has a pinion wheel engaging with a vertical or commonly a horizontal hydraulic rack by means of which the vessel is turned up or down, as was shown in Figure 178. The lining, about 1 foot thick, is at present usually made of refractory sandstone, or blocks of ganister (a siliceous sandstone: SiO_2, 95; $Al_2O_3 + Fe_2O_3$, 4; CaO, MgO, Na_2O, 1), and joined and faced with ganister mixture. In order to allow for the expansion of the sandstone the shell of the converter is plastered with fire clay in which are embedded laths. These char and shrink when the converter is being heated and gradually burn out. Formerly the lining was more frequently rammed, a mixture of quartz and clay being used. A lining lasts about six months, being patched weekly. The lining is slowly dried by burning coal with a gentle blast; the temperature is gradually raised until it has reached a yellow heat, when the vessel is ready for its charge. The bottom is usually made of rammed plastic refractory material (two parts ganister; one fire clay). With the so-called "pin bottom," formerly in use, the mixture was rammed around long iron pins, and these were afterward withdrawn, leaving holes for the blast. In this country blast is admitted through twelve to twenty-five fire-clay tuyères (Figure 195) 20–26 inches long, having seven or nineteen holes $\frac{3}{8}$–$\frac{1}{2}$ inch in diameter, the tuyère area varying from 2.5 to 4.0 square inches per ton of charge.[1] The disposition of the tuyères differs in different works. In Figure 192 is shown that of the Edgar Thomson converter. That of the Maryland Steel Company is represented in Figures 190 and 196–197. Figure 196 shows the position of the tuyères as seen through the mouth of the converter. Long fire-clay tiles, set on end, reach to the bottom, that they may bind the tamping firmly together. Figure 197 shows the tuyères as seen from the wind box. To the tuyère plate is cast a dovetail, into which slips the flange of the set screw e, which holds the tuyères in place. In case a tuyère is eaten out and has to be blanked, a ball of plumbago-clay mixture is rammed in and the opening closed by an iron disk, which is held in place by the set screw e. A bottom has to be dried slowly and thoroughly and then brought to a red heat so that it can quickly replace one that is worn out. The furnace in common use is shown in Figure 198.[2] A = fireplace, B = flue, C = oven, D = roof of oven, E = door of oven, F = bottom car, G = converter bottom, H = flue leading to stack. A bottom lasts for eighteen to twenty-five blows.[3] About twelve bottoms are always kept on hand for one converter. In replacing an old bottom, the bottom car is brought into position, its platform raised to touch the converter bottom, the key bolts holding the bottom shell are loosened, the bottom is pried off and taken to the repair shop. The rim of the converter receives a plumbago wash, and the car, with a heated fresh bottom, is brought into position. Its rim also receives a plumbago wash and then a lute of ganister mud. The bottom is raised and clamped. The exchange takes ten to fifteen minutes.

The Blowing Engines[4] are mostly horizontal and of the Corliss type. They have to be powerful, delivering 8,000–11,000 cubic feet air per minute at a pressure of 25–30 pounds to

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[1] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 157.

[2] Richards, "Notes on Iron," p. 90.

[3] "Transactions American Institute of Mining Engineers," XII, p. 254.

[4] *Op. cit.*, XXII, pp. 537, 709.

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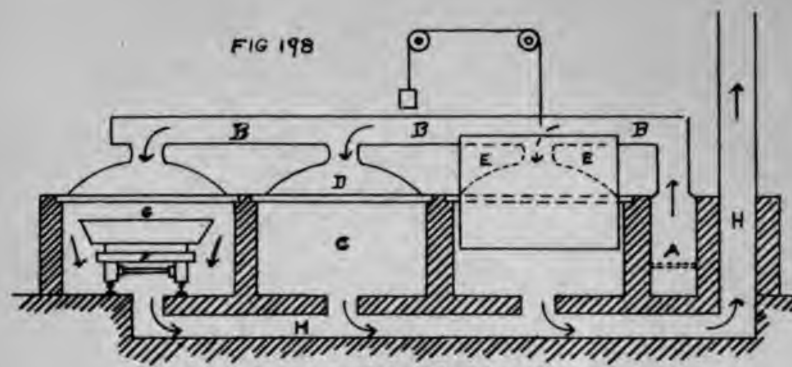
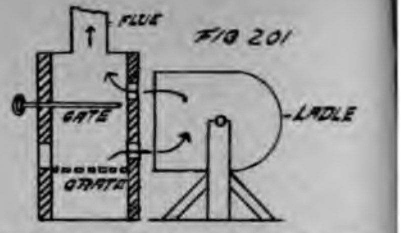
ASTOR, LENOX AND
TILDEN FOUNDATIONS

FIG 198



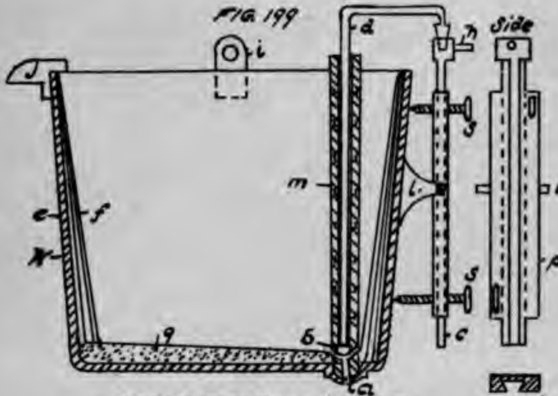
BOTTOM HEATING FURNACE

FIG 201



HEATING STEEL CASTING LADLE

FIG. 199



STEEL CASTING LADLE

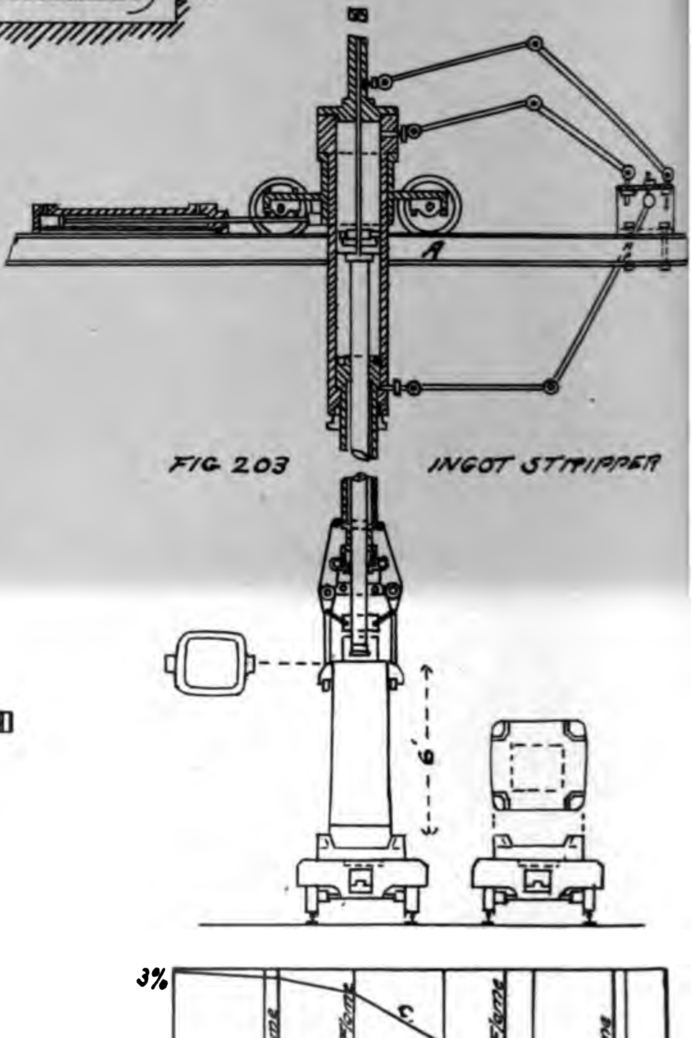
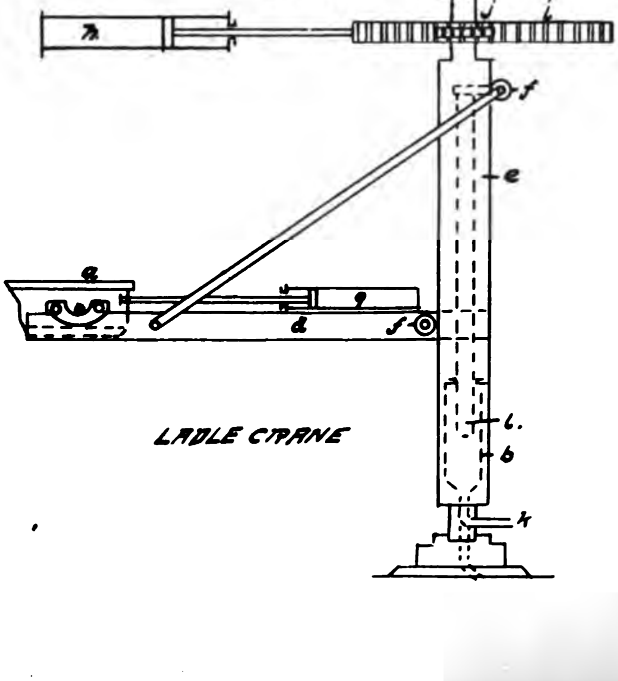


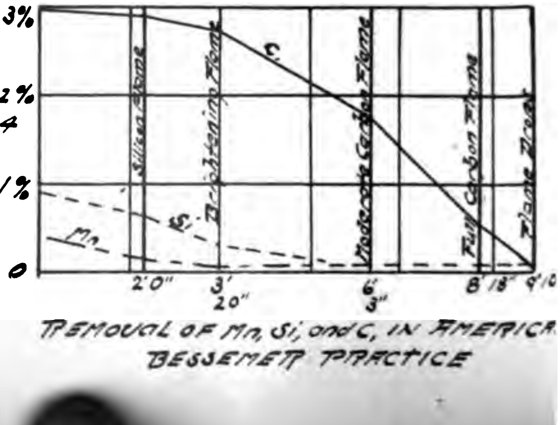
FIG 203 INGOT STRIPPER

FIG. 202



LADLE CRANE

FIG. 204



REMOVAL OF Mn, Si, and C, IN AMERICA
BESSEMER PRACTICE

the square inch. The air valves are of the fan-shaped gridiron type, and are actuated by steam.[1] The South Chicago Works in 1893 had three 10-ton converters which were served by horizontal engines: air cylinder 66 inches diameter, steam cylinder 54 inches, stroke 60 inches, revolutions 21 + per minute, blast pressure 20–25 pounds. The Johnson Company, Lorain, Ohio, in 1897 had two 15-ton converters served by two horizontal engines: air cylinder 60 inches diameter, high-pressure steam cylinder 36 inches, and low-pressure 72 inches diameter, stroke 60 inches, revolutions 35–40 per minute, blast pressure 25 pounds. Hot blast is suggested by Wiborgh.[2]

SECTION 53. **Ladle, Crane, and Mold.** (1) *The Steel Casting Ladle.* Details of a steel casting ladle which is similar to the one shown in Figures 176–178 are given in Figures 199–200. The plate-iron shell *K* is lined with a course of split brick, and upon this is rammed moistened loam, the lining being about $4\frac{1}{2}$ inches thick at the bottom and tapering toward the top. The pouring hole is fitted with a cylindrical nozzle (life, four heats) having an opening $1\frac{1}{2}$–2 inches in diameter, and a bowl-shaped seat for the stopper. This consists of the iron rod *D*, to the lower end of which is fastened, by means of a T-bolt, the plumbago end *B*. The rod is protected by a sleeve *M* made up of annular fire-clay bricks. As it has to be exchanged frequently, say every four casts, it is simply fitted into the socket of the sliding bar *C*. It is essential that the stopper should fit the nozzle closely to prevent any leakage, and must therefore be made adjustable (Holley's device). The sliding bar *C* passes through a grooved sleeve *P* which rests with its trunnions, *L*, in a casting riveted to the ladle. The sleeve is adjusted by the set screws, *S*. The stopper is raised and lowered by means of a lever attached to the top (*F*), or preferably to the bottom of the rod *C*. In order to insure a tight joint, sand is often thrown into the ladle before it is filled. After the steel has been teemed, the ladle is inverted and the slag poured off through the spout *J*. The sketch presupposes the ladle to be suspended by the ear *I*. After lining, the ladle must be slowly dried and well heated, which takes about two to three days. This was formerly done by tilting the ladle over a coke fire, or having a coke fire in it, the air being supplied from a blower. At present it is warmed in the same way as the converter bottom (Figure 198), or in the manner shown in Figure 200; often producer or natural gas is burned in it. A ladle lining lasts about forty heats. The amount of refractories consumed per ton of ingot was, in one instance:[3] quartz 22, clay 16, sand 21, old bricks 3, loam sand 30; total 92 pounds; 0.102 tuyère was used per ton of ingot.

(2) *The Ladle Crane*, like most cranes in a steel plant, is hydraulic, requiring powerful pumps and an accumulator.[4] The Wellman crane has come into general use, replacing the Worthington, or so-called Holley type.[5] The working of the Wellman crane, similar to the one shown in Figures 176–179, is represented diagrammatically in Figure 202. In it $A =$ steel casting ladle, $B =$ raise cylinder, $C =$ top steadiment, $D =$ jib, $E =$ mast, $F =$ rollers, $G =$ racking cylinder, $H =$ swinging cylinder, I and $J =$ rack and pinion, $K =$ three-way cock for water, $L =$ plunger. Electric Cranes: "Iron Age," January 7, 1904. Speed of Cranes: "Iron Trade Review," August 27, 1903.

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[1] Southwark Foundry and Machine Company, Philadelphia.

[2] "Journal Iron and Steel Institute," 1899, I, p. 197.

[3] *Op. cit.*, 1890, II, p. 132.

[4] Classification by Towne: "Transactions American Institute of Mining Engineers," IV, p. 288.

[5] Howe, "Journal Iron and Steel Institute," 1890, II, p. 130.

(3) *The Ingot Mold and Ingot.* Figures 176 and 178 show pairs of ingot molds on cars standing on the track leading to the stripper (hydraulic ingot extractor), which is located outside of the building on the road to the soaking pits or heating furnaces. This method, started by F. W. Wood in 1890, is the one in common use in modern plants. Formerly the ingots were stripped in the converting building, picked up by a crane, deposited on a truck, and taken away by a locomotive. Figure 203[1] represents the Aiken Hydraulic Ingot Extractor[2] above a full ingot track, and the empty ingot track carrying the ingot carriage. The extractor is represented as running on an elevated track *A*, the horizontal hydraulic cylinder *B* serving to move it to and fro. The extractor has a centre ram 7, and outer ram 9, with centre ram 7 having the piston 6. Water is admitted above through pipe 18, and beneath through pipe 20. This is connected by pipe 22 with cylinder 6, and passes through it, sliding in pipe 21 as piston 6 rises and sinks. With outer ram 9 having piston 8, water is admitted below by pipe 19. The centre ram 7 rising, opens by means of toggle-joints, *C*, the links, *D*, suspended from the outer ram, until the collar strikes the under side of 9. Owing to the water between 7 and 9 being confined, water admitted below 8 would raise the two rams together, leaving the links open. By exhausting water from 19, the rams descend until the links are opposite the lugs, *E*, of the mold; by now exhausting water between the two pistons the centre ram sinks and allows the collar to fall with the links against the sides of the mold under the lugs. Pipe 18 being closed and pipe 20 open to the exhaust, water admitted beneath 8 will raise the outer ram 9 carrying yoke and links, while the ingot butted against the inner ram 7 will be forced out. The mold is raised and deposited on the truck standing on the empty mold track. Should the ingot get stuck, water is admitted through 18, which forces out the ingot. A modified form by Evans is described in "The Engineer," 1899, p. 366; "Stahl und Eisen," 1900, p. 211.[3]

Ingot molds[4] are of cast iron. They should be hard, and also able to stand the strains of expansion and contraction without getting brittle. The composition of the iron is Si, 1.70; P, 0.10; S, 0.05; Mn, 0.75. A mold stands about ninety casts.

SECTION 54. **Chemistry,[5] Method of Operating, and Results.** In the acid Bessemer Process, P and S are not eliminated at all, as the slag is too acid. Tamm recorded[6] some experiments in which 1,850 kilograms pig iron were blown down to 600 kilograms steel, samples being taken, A at the dropping of the flame, B two hours later, C at the end of the operation.

	Pig.	A	B	C
C	4.05	0.03	0.025	0.020
Si	1.125	0.025	0.040	0.014
P	0.024	0.029	0.046	0.066
S	Trace	Trace	Trace	Trace
Mn	4.40	0.10	0.04	0.03

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[1] "Iron Age," March 5, 1891.

[2] Fay, Evolution of Ingot Strippers: "Iron Trade Review," May 1, 1902, p. 64.

[3] Older form of ingot pusher, see "Iron Age," April 11, 1889, Jones; February 13, 1890, Wellman; see also "Stahl und Eisen," 1901, pp. 1104 and 1105; "American Manufacturer," 1901, p. 1194, Wood Stripper; Harbord, "Metallurgy of Steel," p. 36.

[4] "Stahl und Eisen," 1903, p. 375; "Iron Age," April 23, 1903.

[5] Åkerman, "Transactions American Institute of Mining Engineers," XXII, pp. 265, 661.

[6] "Jernkontoret's Annaler," 1886, p. 129, through Jüptner, "Compendium," p. 351.

The 1,850 kilograms pig iron (with 0.029 per cent. P) contained 0.444 kilograms P. The 600 kilograms steel (with 0.066 per cent. P) contained 0.396 kilograms P, showing that there was practically no elimination. The non-elimination of S will be shown below.[1]

The rate at which Si, Mn, and C are eliminated in the converter depends mainly on the percentage of Si in the pig iron and the initial temperature.

(1) With a high-silicon (2–3 per cent.) pig iron and a low initial temperature, at first only the Si will burn, and when the iron is heated to a certain point (assisted by the oxidation of Mn), C begins to burn to CO (English Method).[2]

(2) With a medium-silicon (1.3–2 per cent.) pig iron and a high initial temperature, Si, C, and Mn will be oxidized at about the same time, C more completely than Si or Mn (Continental European Method).[3]

(3) With a low-silicon (0.6–1.3 per cent.) pig iron and a high initial temperature, C, Si, and Mn are oxidized at about the same time. When the temperature exceeds a certain limit, C seizes the whole of the O, leaving Si and Mn in the steel (Swedish Method).[4]

(4) With a low-silicon (0.6–1.3 per cent.) pig iron and a low initial temperature, Si will be almost completely eliminated before the C begins to be much oxidized. In order to make this possible, it is essential that there shall be a large output, which means quick blowing and short intervals between blows. This is the American Method.[5]

The following table (see next page) gives the record of a blow made by A. Julian at the South Chicago Works of the Illinois Steel Company in 1890.[6] The chemical changes are graphically represented in Figure 204. The barometric pressure was 28.79 inches, the temperature of the air 36° C., the blast pressure 27 pounds to the square inch.

In operating, the converter is first turned horizontally and the blast shut off. The pig metal is then poured in, the blast turned on, and the converter turned up. The blast rushes with sufficient force through the tuyère holes to prevent any metal from entering. Si begins to burn, but hardly any flame issues from the nose of the converter, as most of the O is taken up by the Si and the rest by the iron, which forms a slag with the SiO_2, and only N passes off. Gr. C. is changed into C. C.[7] After two to three minutes, when the Si has nearly all been burned and the temperature of the metal bath sufficiently raised, C begins to burn to CO and an orange-colored flame appears, throwing out sparks of burning iron and particles of slag. The flame increases in length and becomes white; the O of the air as well as that of the FeO acts violently upon the C, causing a rumbling noise like distant thunder. About eight to nine minutes after starting, when the carbon has been reduced to 0.04 per cent., the flame suddenly drops and an instant later becomes brownish from Fe_2O_3. The blow is finished, the converter turned down, the spiegeleisen (10 per cent.) poured in (or the heated

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[1] Howe's Table of American Practice.

[2] Diagram, Ledebur, "Manual," 4th ed., p. 1000.

[3] Diagram, *op. cit.*, p. 1002.

[4] See also Åkerman, "Transactions American Institute of Mining Engineers," XXII, p. 265; Diagram, Ledebur, "Manual," p. 1003.

[5] Diagram, Figure 204.

[6] Howe, "Journal Iron and Steel Institute," 1890, II, p. 102; "Transactions American Institute of Mining Engineers," XIX, p. 1127.

[7] Carulla, "Journal Iron and Steel Institute," May, 1904.

ferromanganese, 0.8–1.0 per cent., shoveled in), and the heat teemed. Most of the slag remains in the converter unless it is very thin from having much Mn (and perhaps Al_2O_3) or from being very hot. It is dumped into a cinder car standing under the converter. The contents of the ladle are now tapped into the ingot molds (teemed) and the slag remaining in it dumped into a car.

METAL.

	INITIAL CHARGE.			TIME OF ACTUAL BLOWING.						
	Molten metal.	Steel scrap, estimated.	Average.	2 min. 0 sec.	3 min. 20 sec.	6 min. 3 sec.	8 min. 8 sec.	9 min. 10 sec.	9 min. 20 sec. *A.*	After adding spiegeleisen.
C	3.10	0.36	2.98	2.94	2.71	1.72	0.53	0.04	4.64	0.45
Si	0.98	0.08	0.94	0.63	0.33	0.03	0.03	0.02	0.35	0.038
Mn	0.40	0.97	1.43	0.09	0.04	0.03	0.01	0.01	14.90	1.15
P	0.101	0.10	...	0.104	0.106	0.106	0.017	0.018	0.139	0.109
S	0.06	0.08	0.06	0.06	0.06	0.06	0.06	0.06	...	0.059

SLAG.

SiO_2	...	...	...	42.40	50.26	62.54	63.56	...	...	62.20
Al_2O_3	...	...	...	5.63	5.13	4.06	3.01	...	...	2.76
FeO	...	...	...	40.29	34.24	21.26	21.39	...	...	17.44
Fe_2O_3	...	...	...	4.31	0.96	1.93	2.63	...	...	2.90
MnO	...	...	...	6.54	7.90	8.79	8.88	...	...	13.72
CaO	...	...	...	1.22	0.91	0.88	0.90	...	...	0.87
MgO	...	...	...	0.36	0.34	0.34	0.36	...	...	0.29
P	...	...	...	0.008	0.008	0.060	0.014	...	...	0.010
S	...	...	...	0.009	0.009	0.014	0.008	...	...	0.011
Appearance of flame	...	...	...	Silicon flame	Brightening	Moderate carbon flame	Full carbon flame	Flame drops	Blown 10 sec. after drop	...
Cubic ft. of air	...	...	...	34,502	30,628	53,481	42,265	26,430	1,868	...
Cubic ft. of air per min., *B.*	...	...	...	17,251	22,971	19,691	21,810	25,685	11,208	...

A. Sample V was taken after blowing nine minutes and ten seconds. The vessel was then turned up again and blowing was continued for ten seconds more, when the spiegeleisen was added, no sample of metal or slag being taken before adding it. After adding it, sample VI of both slag and metal was taken.

B. The volume of blast is calculated from the volume of the blast cylinders without deducting for leakage or clearance. Each of the fourteen tuyères has twelve holes. Those in eleven of the tuyères are ⅝ inch in diameter; those in the remaining three tuyères are ½ inch in diameter.

RECORD.

Date 1890.	Turn.	No. of heat.	CONVERTER RECORD.							RECORD OF BLOW.			
			Molten metal.	Steel scrap before blow.	Steam equal to scrap.	Spiegeleisen.	No. of heat on bottom.	Tuyères.		Temperature.	Rate of flowing.	Turned down.	Spiegel re-action.
								No. ⅜ inch	No. ½ inch				
August 13	Day	1,839	21,500	1,000	800	2,500	4th	11	3	Medium	Quick	Fast	Medium

Date 1890.	Turn.	No. of heat.	LADLE RECORD.						INGOT RECORD.					Blown record.	RAIL RECORD.		
			Rate of flowing.	Nozzle.		Behavior in ladle.	Behavior in moldings.	Temperature of molds.	No. of ingots cast.	Kind of ingots.	Bend test.	Time in pit.			Weight per yard.	No. rolled.	Seconds, %.
				Size.	Heat on nozzle.							Hours.	Minutes.				
Aug. 13	Day	1,839	Fast	1¾ in.	3d	Quiet	Quiet	Warm	7⅔	80 lb.	Left R. A.	1	23	Clean without cracks	80 lbs.	23	0

If the metal in the converter is too hot, it is cooled by "scrapping," *i. e.*, charging cold steel scrap, which cools the metal and, running low in Si, reduces the percentage of Si. If not enough scrap is to be had, the metal is cooled by admitting steam with the blast. If the metal is too cool, it is heated by partly turning the converter so that one or more tuyères shall be bared from metal, when the air blowing through them over the metal will burn the CO to CO_2 and at the same time oxidize some Fe. Thus the metal is quickly brought to the desired temperature. In order to heat uniformly, the converter is seesawed, and thus some tuyères are bared first on one side and then on the other.

The progress of a heat was formerly watched by means of the spectroscope;[1] today it is done with the eye either protected by blue glasses or by means of so-called Bessemer spectacles, which show a green light during the time that the C is burning. These spectacles are made up of three lenses, one orange (cutting out chiefly blue and violet), one brown smoked glass (reducing the intensity of the light), and one a peculiar blue (cutting out chiefly yellow and orange and some red). They are cemented together with Canada balsam and then ground to fit into the frame. The Bessemer spectrum[2] shows banded lines of blue and green; blue disappears and is followed by the green when the C is burned out. By the spectacles all colors, excepting green with a small seam of red, are cut out.

The two following records of blows were made in 1899 at the Jones and Laughlin Works, Pittsburgh, Pennsylvania, and at the Carnegie (formerly Edgar Thomson) Works, Braddock, Pennsylvania.

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[1] Wedding, "Das Schmiedbare Eisen," p. 400; Hartley, "Journal Iron and Steel Institute," 1895, II, p. 95.

[2] Schellen, translated by Lassel-Abney, "Spectrum Analysis," Longmans, Green & Company, London, 1885, p. 257; or Roscoe, "Spectrum Analysis," Macmillan, London, 1885, p. 200.

EIGHT–TON ECCENTRIC CONVERTER.

Hour.	Minute.	Second.	Observations.	Hour.	Minute.	Second.	Observations.
10	29	20	Metal poured into converter.	10	32	15	C-flame begins to show.
10	29	40	Pouring finished.	10	33	00	Full C-flame.
10	30	00	Metal-pouring ladle moved away.	10	38	05	Converter turned down.
10	30	15	Blast turned on.	10	38	40	Blast turned off.
10	30	30	Converter turned up and Si-flame appears.	10	48	00	Steel poured and ferromanganese shoveled into ladle while pouring.

Hour.	Minute.	Second.	Observations.
10	49	30	Pouring finished, *i. e.*, the slag has been discharged and the converter is again in position ready for a new charge.

Time between heats, 20 minutes, 10 seconds.	Time of blowing, 8 minutes, 25 seconds.

FIFTEEN–TON CONCENTRIC CONVERTER (FIGURES 179, 191–194).

33,000 POUNDS DIRECT METAL, 2,500 POUNDS SCRAP, 3,000 POUNDS SPIEGEL.

Hour.	Minute.	Second.	Observations.	Hour.	Minute.	Second.	Observations.
10	01	10	Metal poured into converter.	10	16	30	Green disappears, converter turned down, blast shut off, spiegeleisen poured into converter and contents into steel ladle, slag overflowing at the back.
10	02	40	Pouring finished.				
10	02	45	Blast turned on, converter turned up, Si-flame appears, scrap fed.				
10	06	20	C-flame appears; glasses show a greenish tinge.	10	18	00	Pouring finished, converter turned up and back.
10	15	40	Green begins to fade.	10	18	20	Converter ready for new charge.

Time between heats, 17 minutes, 10 seconds.	Time of blowing, 13 minutes, 45 seconds.

The amount of slag formed is about 12 per cent. on the weight of the pig iron.[1] The loss in iron in the converter is about 10 per cent. Of the Mn added in the spiegeleisen 70 per cent. enters the steel, while 30 per cent. is oxidized and enters the slag; of the C added by the spiegeleisen 80 per cent. enters the steel.[2]

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[1] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 165.

[2] Richards, "Notes on Iron," p. 87; also Campbell, *op. cit.*, pp. 165, 174. Calculation of amount of spiegeleisen required: Carulla, "Journal Iron and Steel Institute," May, 1904.

SECTION 55. **Heat Evolution.** J. W. Richards[1] gives the following calculation of the rise of temperature in Bessemerizing. It is based on 1,000 kilograms pig iron (Si, 1 per cent.; C, 3.50 per cent.) and the supposition that the initial temperature is 1,400° C. and the average temperature 1,600° C. There is a loss in weight of 8 per cent., and one of 2 per cent. of iron which is burned. At 1,600° C. the specific heats of the gases are per cubic meter: CO = 0.40, N = 0.40, CO_2 = 1.34; per kilogram: CO = 0.32, N = 0.32, CO_2 = 0.68;[2] of the metal products per kilogram: steel, 0.21; slag, 0.25; of the lining, 0.25.[3] The specific heat of air between 100° C. and 1,400° C. is per cubic meter, 0.346; per kilogram, 0.268.

NET HEAT DEVELOPMENT.

	Calories.	Surplus.
Combustion of Si:—		
10 kg. Si + 11.4 kg. O = 21.4 kg. SiO_2 =	64,140	
11 kg. O = 49.6 kg. air absorbing 49.6 × 0.268 × 1,300 =	17,280	
		46,860
Combustion of Fe:—		
20 kg. Fe + 5.7 kg. O = 25.7 kg. FeO =	23,460	
5.7 kg. O = 24.8 kg. air absorbing 24.8 × 0.268 × 1,300 =	8,640	
		14,820
Combustion of C:—		
7 kg. C + 18.7 kg. O = 25.7 kg. CO_2 =	56,930	
28 kg. C + 37.3 kg. O = 65.3 kg. CO =	68,600	
	125,530	
56 kg. O = 243.5 kg. air absorbing 243.5 × 0.268 × 1,300 =	84,830	
		40,700
Total surplus heat developed		102,380

CALORIFIC CAPACITY OF THE PRODUCTS.

Weight × specific heat at 1,600° C.

920 kg. liquid steel	× 0.21	= 193.2
150 kg. liquid slag	× 0.25	= 37.5
50 kg. lining	× 0.25	= 12.5
25.7 kg. CO_2	× 0.68 × ¾	= 13.1
65.3 kg. CO	× 0.32 × ¾	= 15.7
244.8 kg. N	× 0.32 × ½	= 39.2
Total capacity per 1° C.		= 311.2

$$\text{Theoretical rise of temperature} = \frac{102{,}380}{311.2} = 329^\circ \text{ C.}$$

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[1] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 166.

[2] As the gases pass off continuously, it is assumed that three-fourths of the CO and CO_2 and one-half of the N take part in the total rise of temperature.

[3] Assuming that 1 centimeter of it participates in the increase of temperature.

Starting with 1,400° C., the end temperature would be $1,400 + 329 = 1,729°$ C. Assuming the loss by radiation to cause a fall of 50° C., the total rise would be 279° C. and the end temperature 1,679° C. See also Hartley, "Journal Iron and Steel Institute," 1895, II, p. 95.

Le Chatelier[1] measured the temperature at the end of a blow in a Robert-Bessemer converter and found it to be 1,580° C. When the C began to burn, the temperature was 1,330° C.; the temperature of the melted steel, 1,450° C.

SECTION 56. **Products.** The products of the converter are ingot steel, slag, gases, fumes, and odds and ends.

(1) *Ingot Steel.* The form of the ingots is that of a pointed truncated four-sided pyramid (see Figure 203). The dimensions of an ingot vary with the weight desired, thus: an ingot 16½ x 18 inches x 7 feet = 5,000 pounds. Harbord[2] gives tables of sizes of ingots and their weights. The composition of Bessemer steel varies greatly according to the use to which it is to be put, the leading variables being C and Mn; soft steel: C, 0.08–0.10 (lowest, 0.03); P, 0.08–0.10 (lowest, 0.04); Mn, 0.45–0.50; S, 0.04–0.06; Si, none. Hard steel: C, 0.40–0.60; P, 0.08–0.10; Mn, 0.75–0.90; S, 0.05–0.10; Si, 0.05–0.15.

Uneven Composition of Ingots.[3] When the liquid steel is poured[4] into the thick cast-iron mold, the parts in contact with the mold solidify quickly, forming a solid envelope, enclosing a liquid centre. In the subsequent slower cooling process, segregation takes place, which disturbs the former homogeneity of the fluid steel. Steel may be considered as a dilute solution of C, P, S, Si, Mn, etc., in iron. Upon cooling, there will be a tendency for the solvent, *i. e.*, pure iron, to crystallize out first, to be followed by the other elements. The order in which these separate out is, according to Pourcel,[5] approximately C, P, S, Si, and Mn; Cu, if present to any extent, separates rapidly unless prevented by the presence of Al; Ni hardly separates at all. The impurities form new alloys, which, being lighter and more fusible than the crystallizing iron, rise to the surface and collect in the top of an ingot. Thus Snelus[6] cast an ingot 19 x 19 x 84 inches, allowed it to cool very slowly, took top (21 inches below) and bottom (4 inches above) samples, which upon analysis gave: —

	C. C.	Si	S	P	Mn	Fe
Top	0.760	Trace	0.187	0.191	0.558	98.304
Bottom . . .	0.350	Trace	0.044	0.044	0.514	99.038

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[1] Howe, "Engineering and Mining Journal," October 11, 1890, p. 428.

[2] "Steel," pp. 31–34.

[3] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 340.

[4] Casperrson, Influence of Pouring Temperature: "Stahl und Eisen," 1883, p. 71.

[5] "Transactions American Institute of Mining Engineers," XXII, p. 107.

[6] "Journal Iron and Steel Institute," 1881, II, p. 379.

showing how especially the metalloids collected near the top. He further analyzed six top and bottom samples taken diagonally from the corner of the sample plate to the centre:—

	Top.			Bottom.		
	C	S	P	C	S	P
No. 1 corner .	0.44	0.032	0.044	0.44	0.048	0.060
No. 2 . . .	0.54	0.048	0.060	0.42	0.056	0.062
No. 3 . . .	0.57	0.080	0.086	0.41	0.048	0.054
No. 4 . . .	0.61	0.096	0.097	0.40	0.048	0.054
No. 5 . . .	0.68	0.120	0.111	0.38	0.048	0.054
No. 6 centre .	0.77	0.187	0.142	0.37	0.044	0.052

They show that differences between the samples from corner to centre at top are greater than those at bottom. See also Stubbs[1] and Weston.[2]

Soundness of Ingot. If an ingot is allowed to cool and then cut lengthwise, the surfaces are liable to show small blowholes, and toward the centre may be found a large one, forming a "pipe" (Figures 205–208). The cavities are due to the shrinkage of the metal and to the presence of gases. When the hot metal is cast into the cold mold an envelope of solid metal forms, surrounding the liquid metal in the centre. The solid crust, while still shrinking during the cooling, does not contract as much as the liquid metal, which settles on the solid crust, contracts (contraction coefficient $= \frac{1}{60} - \frac{1}{72}$), and finally leaves an empty space, a pipe, where the metal solidified last, *i.e.*, near the top, if the ingot, as is usual, cools in an upright position. If an iron mold were used, lined in part with badly conducting brickwork, as shown in Figure 209, the steel in proximity to the brickwork would remain liquid longer than the rest, and the cavity formed, not being filled from above, would form a pipe *b* in addition to the normal pipe *c*. A contracted mold (Figure 210) would have a similar effect, as the quickly cooling narrow section would prevent the flow of metal from the ends to fill up the pipes. In order that the pipe may injure as little of the ingot as possible, it should be near the top, which necessitates keeping this as long as possible from solidifying. It is done by covering the liquid steel in the mold with sand and then wedging fast an iron cover. The top layer of steel will in part solidify, and, being covered with a bad conductor of heat, will retard the loss of heat by radiation and keep the steel below liquid, at least for a while. This will refill any cavity that might form lower down. (Sinking head.)[3] This "stopping" is also efficacious in preventing the steel from boiling over, characteristic for dead-soft steel. De Laval's feeding head[4] is similar in effect to the sinking head. Fluid compression[5] of the ingot during solidification has given satisfaction

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[1] "Transactions American Institute of Mining Engineers," XXII, p. 107.

[2] "Iron Age," May 19, 1904, p. 28.

[3] Howe, "Steel," p. 153.

[4] "Journal Iron and Steel Institute," 1902, I, p. 336; 1904, May meeting (Johns).

[5] Howe, "Steel," p. 153; Harbord, *op. cit.*, p. 495.

in many instances. Thus Krupp[1] employed liquid carbonic acid; Witworth[2] and Harmet[3] hydraulic compression. While Witworth applied the pressure from above, and thus aimed at preventing the steel from giving off gas, Harmet applied it from below, the tapering mold having parallel sides for a short distance from the bottom to receive the piston, and closed up the pipe. Fluid compression is of no value with small ingots, and with large ingots the pressure exerted[4] must be at least 2 tons per square inch. Centrifugal casting has been suggested by Sebenius;[5] Stridsberg rotates the ingot along its longitudinal axis; Goldschmidt[6] proposes to use thermite (anti-piping thermite) to keep the top of the ingot liquid; Sauveur-Whiting[7] have experimented with an overflow method to prevent piping. The blowholes found in the ingot are due principally to the escaping of H and N held in solution by the steel, although they are caused to some extent by CO held partly in solution and resulting partly from the chemical reaction of iron oxide upon carbon. The evolution of CO causes the steel to give off H abundantly. The capacity of steel to absorb gases increases with the temperature. As the metal cools, the dissolved gases pass off more or less quickly as long as the top of the ingot remains liquid. When this solidifies and the crust becomes sufficiently strong to resist the pressure of the rising gases, these cannot pass off, but fill the spaces left by the shrinkage of the metal inside the liquid shell. The blowholes near the mold form lenticular cavities with their axes at right angles to the mold; those near the centre are less regular in shape and position (Figures 206–208; see also Howe, "Steel," p. 146; Ledebur, "Eisenhüttenkunde," 4th ed., p. 916). This giving off of gases causes the steel to rise in the mold during solidification, the CO burning with a bluish flame. Bessemer steel gives off more gas than open-hearth or crucible steel. Figure 205 represents an ingot (with concave surface and pipe) which has solidified without any gas being given off. Figure 206 forms a transition between Figures 205 and 207; the surface is level, perhaps slightly convex, bubbles are seen at the sides, and the single pipe of Figure 205 is replaced by a number of bubbles. Figure 207: the convex parts of the top overbalance those that are level; blowholes are found covering the entire surface of the ingot. They were formed in the parts that solidified first, and while the top of the ingot was still liquid, or only slightly overcrusted. The rest, which solidified after a heavy crust had been formed on top, did not give off any more gas. Figure 208: the top is very uneven, and does not rise above the mold; there are no bubbles at the surface, they are found halfway between the sides and the centre. When the mold was filled, or while it was filling, gases were set free where the steel came in contact with the mold; the gases wandered toward the fluid centre and upward, but the spaces they occupied were filled directly with liquid metal. During cooling, the top of the ingot solidified and prevented a further liberation of gas. The gas bubbles were arrested in their inward and upward passage, and formed the unsound ingot. One can easily see how forms resembling

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[1] "Stahl und Eisen," 1882, p. 161.

[2] Bethlehem Steel Company, "Cosmopolitan Magazine," February, 1900; Greenwood, "Proceedings English Institute of Civil Engineers," 1889, XCVIII, p. 108; also "Steel and Iron," p. 510.

[3] "Journal Iron and Steel Institute," 1902, II, p. 146; "Bulletin de l'Industrie Minérale," 1902, I, p. 901 (excellent illustrations); "Stahl und Eisen," 1902, p. 123; 1903, p. 1196 ("Iron Age," December 24, 1903); 1904, p. 892.

[4] Harbord, "Steel," p. 498.

[5] "Transactions American Institute of Mining Engineers," XXII, p. 671.

[6] "Electro-chemical Industry," 1903, I, p. 534.

[7] "Proceedings American Society for Testing Materials," 1903, III, p. 129.

combinations of Figures 207 and 208 can arise, and how an ingot can become permeated by blowholes. As the rate of cooling plays such an important part in the soundness of an ingot, it is clear that there must be differences in the pouring temperatures of large- medium- and small-size ingots. The dissolving power of steel for gases is governed, beside the temperature, by the chemical composition; thus Si, Mn, and Al not only diminish the dissolving power, but retard the giving off of H, and, reducing FeO, prevent its reaction upon C. Si is five times as effective as Mn, and Al more effective than Si (Brinell's Density-Quotient).[1] However, only very small amounts of these three elements can remain in the steel without having a harmful effect upon its tensile strength and malleability. The remedies, beside suitable chemical composition and pouring temperature, are to make large castings, to make bottom castings, and to allow the metal to solidify under high pressure.

In large castings, which cool slowly, the gases are given better opportunity to escape than in small ones. In bottom-casting, used only with open-hearth steel (see later), the metal, rising in the mold, is not spattered (causing blowholes) and no air is drawn in, which would aid (as it does in top-casting) in the passing off of gas bubbles on the quickly solidifying envelope of steel.

(2) *Slag.* The table of Howe, page 143, showed the final slag to have the composition: SiO_2, 42.20; FeO, 17.44; Fe_2O_3, 2.90; MnO, 13.72. Campbell[2] gives as an average of 100 heats at Steelton, Pennsylvania: SiO_2, 59.70; FeO, 19.30 (all the Fe being calculated as FeO); MnO, 12.00. Shots of iron are removed with a magnet after crushing the slag.

(3) *Gases.* The gases at first contain about 90 per cent. N, traces of CO, much CO_2, and some free O; with increase of temperature, CO_2 decreases and CO increases, the N being reduced to as much as 65 per cent. H, absent at first, makes its appearance with increase of temperature. The subjoined analyses were made at the works of the Maryland Steel Company, Sparrow's Point, Maryland.[3] See Bell,[4] Snelus,[5] Hartley,[6] Tamm.[7]

Time when taken.	CO_2	CO	O	N	Remarks.
Five minutes from start .	4.6	15.4	Nil	80.0	Pressure of gas in converter, 5 feet from nose, was 4.4 ounces.
Eight minutes from start .	6.4	31.4	Nil	62.2	
End of blow	8.8	35.4	Nil	55.8	

(4) ***Fumes.*** An analysis by Brusewitz[8] of fumes showed SiO_2, 17.92; Al_2O_3, 2.08; Fe_2O_3, 15.55; MnO (probably as Mn_3O_4), 61.61; MgO, 0.25; CaO, 0.61.

[1] "Journal Iron and Steel Institute," 1902, I, p. 347.

[2] "Manufacture and Properties of Iron and Steel," 1903, p. 163.

[3] Private Notes, 1901.

[4] "Principles of Manufacture of Iron and Steel," p. 390.

[5] "Journal Iron and Steel Institute," 1871, p. 247.

[6] *Op. cit.*, 1895, II, p. 109.

[7] "Iron," XIII, pp. 674, 707, 739, 771, 803; XIV, pp. 2, 67, 98, 131.

[8] Jüptner, "Compendium," p. 359.

VERTICAL SOAKING FURNACE HEATED WITH PRODUCER GAS

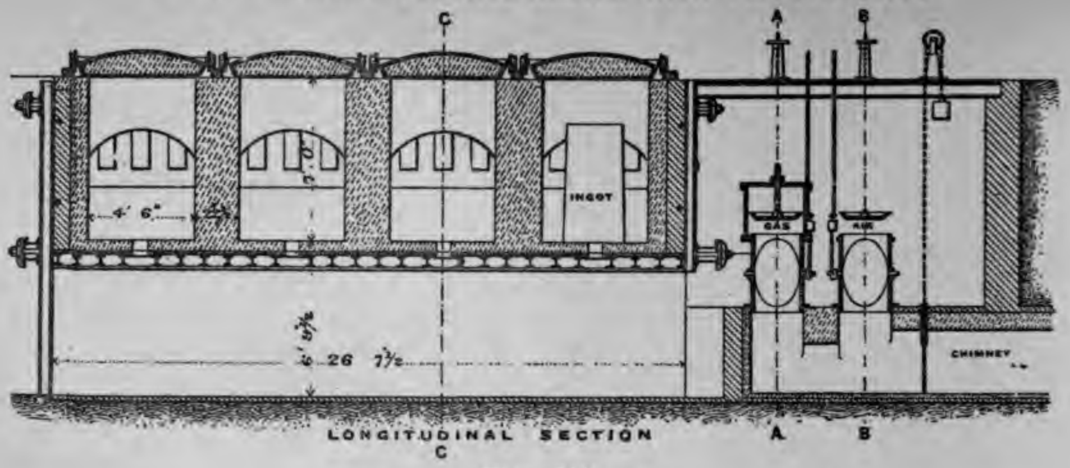

Fig. 213.

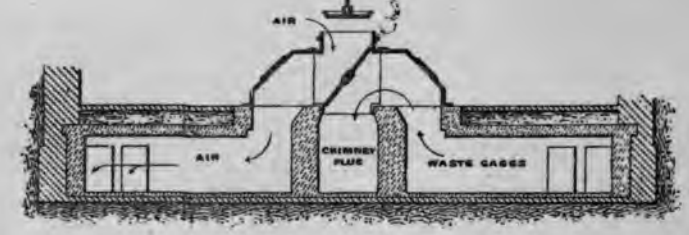

Fig. 214. Section on BB.

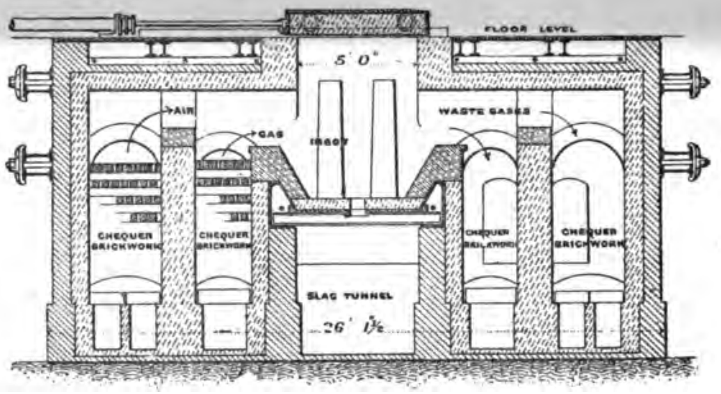

Fig. 215. Section CC.

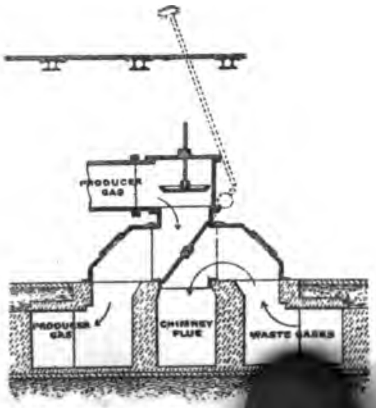

Fig. 216. Section

chains m and closes the jaws n' and n^3 upon the ingot. By raising the jib the jaws bite upon the ingot and clasp it firmly and raise it.

In 1882 Gjers[1] brought out his soaking pits (Figure 212), a number of upright chambers built in a mass of brickwork sunk in the ground below the level of the floor. In them the temperatures of an ingot are equalized. The pits are first heated by placing in them red-hot ingots, but when once heated they require no further warming, as each newly poured ingot has stored in it more heat than is necessary for rolling. These pits are most satisfactory in large plants, having a minimum capacity of 300 tons in twenty-four hours,[2] and working with sufficient regularity to allow exchanging the ingots at short enough intervals to prevent their becoming too cool. Where this frequency and regularity are not obtainable, which is the case in most plants, and pits have to stand empty occasionally, they have to be heated again before using. External heating has to be applied also with small ingots, say under 12 inches square,[3] where the heat stored in the centre is not sufficient to bring the cooler outside to the rolling temperature. The loss of steel by scaling in non-heated soaking pits is 1 per cent.; in heated soaking furnaces 1½–1¾ per cent.; in horizontal heating furnaces 2½–3 per cent.[4]

In Figures 213–216 is represented an ingot-soaking furnace with regenerative chambers heated with producer gas.[5] In the furnace there are four pits (5 feet x 4 feet 6 inches and 7 feet deep), each of which takes two ingots. The pits are closed with covers on wheels drawn aside by hydraulic cylinders. Beneath the floor (brickwork resting on I-beams) is the slag tunnel receiving the slag formed by the iron scale slagging the brickwork. Figures 214–216 show the positions of the two butterfly valves set on air and gas respectively, while the waste gases pass off to the chimney flue. Low-C steel requires less time to become uniformly heated than high-C steel. When the temperature of the pit is high enough not to require any heating, the gas is, of course, simply shut off.

Other heating pits are those of Lareau;[6] Miller;[7] Barrow, England;[8] Wellman-Daniels;[9] Nau.[10]

After the temperature of an ingot has become uniform in a soaking pit or a soaking furnace, it is taken up by a crane[11] straight to the blooming mill, which rolls (roughens) the ingot down to a bloom 7 x 7 inches, which, after having the ragged ends cut off by hydraulic or steam shears,[12] goes either straight to the rail mill or first to a reheating furnace.[13]

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[1] "Journal Iron and Steel Institute," 1882, II, p. 565; "Stahl und Eisen," 1889, p. 916.

[2] "Stahl und Eisen," 1896, p. 61.

[3] *Ibidem.*

[4] Daelen, "Stahl und Eisen," 1896, p. 61.

[5] Head, "Iron Age," December 24, 1895.

[6] Shenango Valley Steel Company: "Iron Age," October 13, 1892.

[7] "American Manufacturer," August 27, 1897.

[8] "Journal Iron and Steel Institute," 1901, I, plate 27.

[9] "Iron Age," June 6, 1901; "Stahl und Eisen," 1902, p. 416.

[10] "Iron Age," October 2, 1902; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1903, p. 86; "Zeitschrift des Vereins deutscher Ingenieure," 1902, XXXVI, p. 1986.

[11] Morgan Overhead Traveling Crane: "Iron Age," September 15, 1892.

[12] Homestead 3,000-ton Shear: "Iron Age," October 18, 1888. Leechburg Foundry and Machine Company, Hydraulic Bloom Shear: "Iron Age," July 24, 1890. Fort Payne Bloom Shear: "Iron Age," August 21, 1890; Daelen, "Stahl und Eisen," 1889, p. 23. Pusey and Jones Company: "Iron Age," June 14, 1888.

[13] Bloom, Billet, Bar, or Slab: "Iron Age," April 12, 1888.

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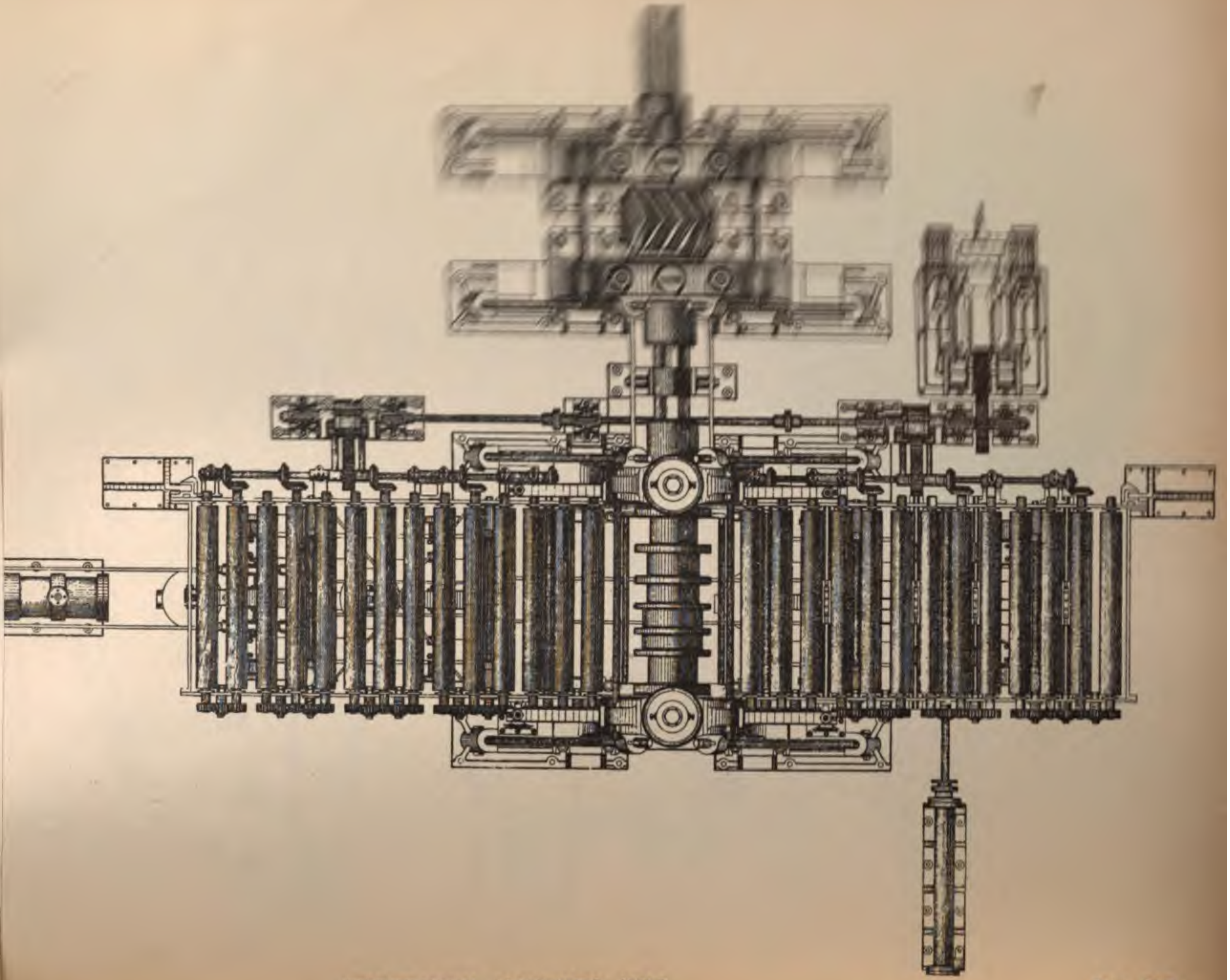

THREE-HIGH BLOOMING MILL.

Fig. 219.

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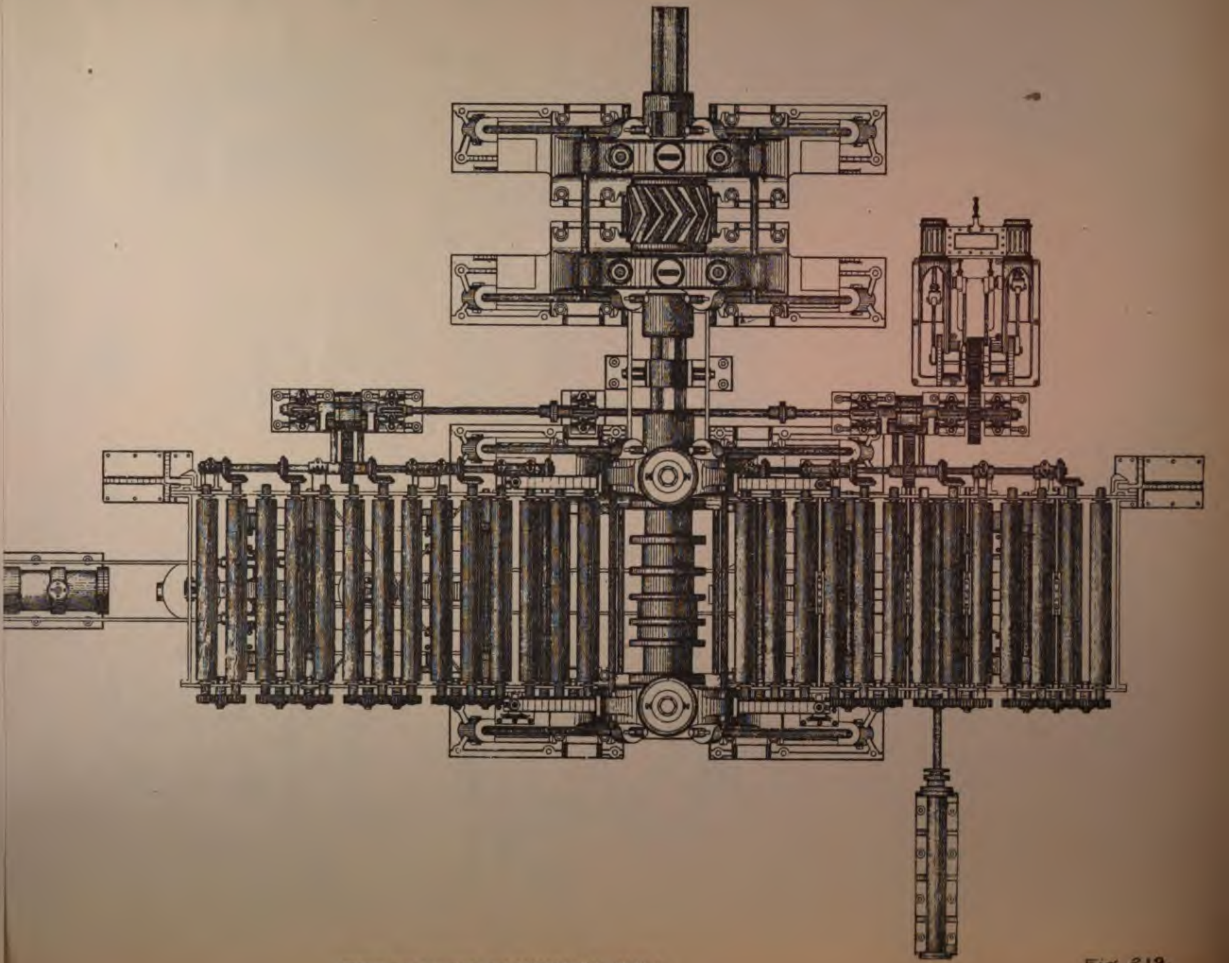

THREE-HIGH BLOOMING MILL.

Fig. 219.

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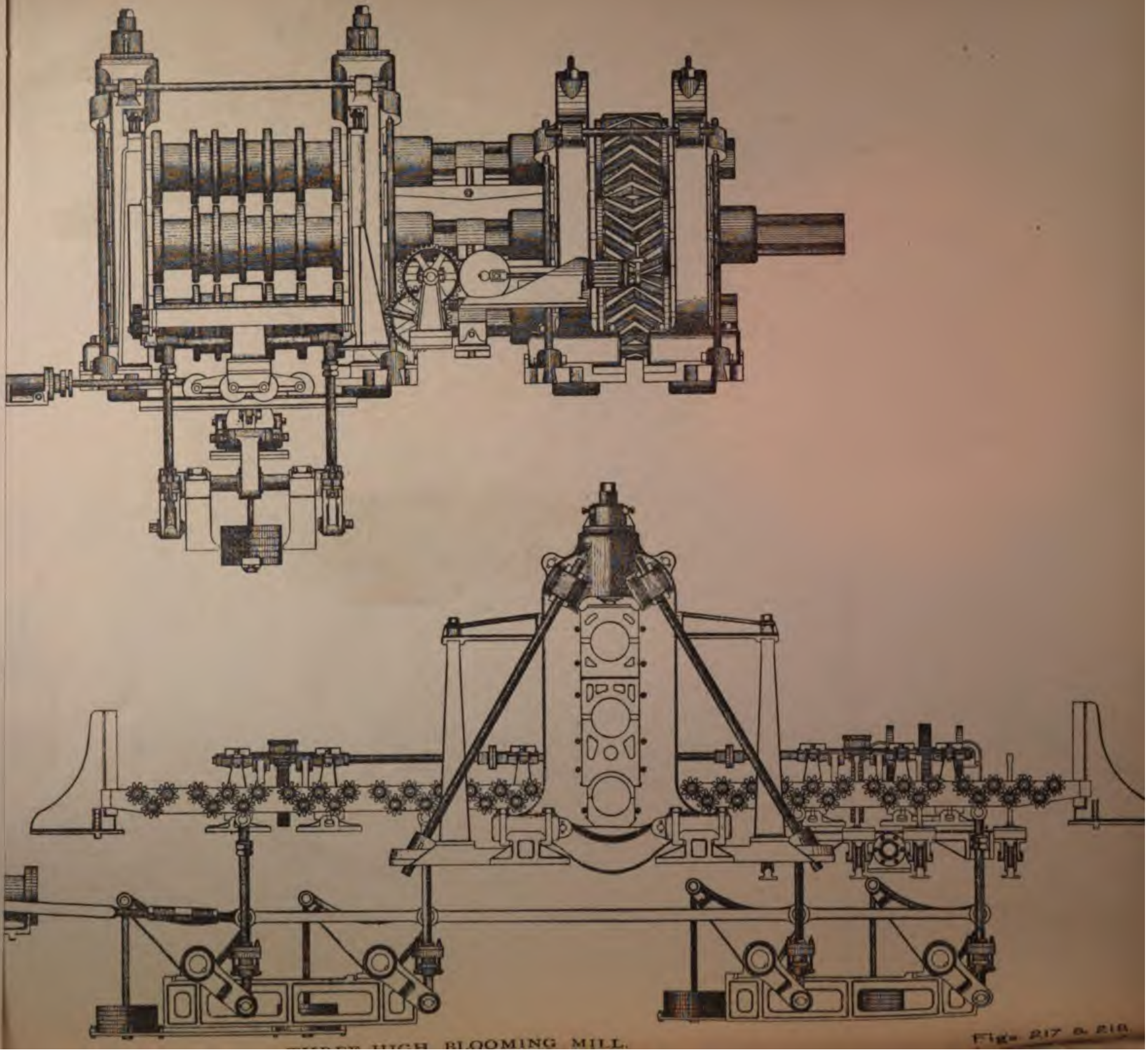

THREE HIGH BLOOMING MILL.

Figs. 217 & 218.

Blooming mills are either two-high reversing mills or three-high non-reversing, with fixed, non-adjustable rolls. With a large quantity of blooms of one size, the three-high blooming mills have the record for the greatest output.[1] Where a blooming mill has to make all sizes of blooms and slabs, the two-high mill is undoubtedly preferable.[2]

The *Three-high 36-inch Blooming* shown in Figures 217–219 is one built by Mackintosh, Hemphill and Company, Pittsburgh, Pennsylvania. The rolls are fixed, therefore the mill produces blooms of only one size. On either side of the rolls are lifting tables with live rollers and ingot manipulators, so placed that the ingot in its twelve passes to and fro shall be brought to the level of the grooves and in front of them at the proper moment. The power (front elevation, Figure 217) is furnished through the central roll and transmitted to the upper and lower rolls by spur gearing with herringbone or double-helical teeth. The manner of raising and lowering the lifting tables through the steam or hydraulic cylinder on the left by means of connecting rods and two pairs of bell-crank levers (with counterweights) is clearly shown in the side elevation (Figure 218). The motion of the live rolls is seen in Figure 219 (plan) and Figure 218. In Figure 219 the main shaft, by means of bevel gearing, sets nine rollers going, each of which communicates the rotary motion to two or five rollers on the opposite side (Figure 218). In Figure 217 is shown the manipulator, having four fingers (Figure 218), with the stationary steam (or hydraulic) piston at the left (see also Figure 219) which moves the carriage to and fro. Figures 218 and 219 further show guide plates for the lifting tables. A perspective drawing of a three-high mill built by the Philadelphia Roll and Machine Company, Philadelphia, Pennsylvania, is published in "Iron Age," April 9, 1903, p. 22.

The *Heron Manipulator* shown in Figure 220 has, beside the usual lateral movement effected here by the upper hydraulic cylinder *a*, an up-and-down movement through the lower hydraulic cylinder *b*. The piston of this forms the water-supply pipe for the vertical hydraulic cylinder *c*, carrying a crosshead (four fingers *e*). As the carriage is moved to and fro by the upper cylinder *a*, the pipe travels in and out of the lower cylinder *b*, and keeps the vertical cylindrical ways connected with hydraulic water.

In the *Critchow Ingot Manipulator* (Figure 221)[3] the "finger" or "go-devil" G is raised and lowered by rack and pinion E, operated by the hydraulic elevating cylinder A; carriage F is manipulated by the hydraulic transverse cylinder B. $C =$ live roller, $D =$ ingot. The turning of the ingot while passing from one set of rolls to another is sometimes accomplished by *tumbling guides*, shown in Figure 222. Other manipulators, see Harbord,[4] Sharon Steel Company (3-inch blooming mill),[5] Differdingen (common in United States).[6]

A description of a two-high mill will be given with rolling plate from open-hearth ingots.

Rail Mill of the South Chicago Works, United States Steel Company (Figures 223–224).[7] The ingots (16 x 16 inches) from the converter go into soaking furnaces of the Siemens type

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[1] South Chicago and Edgar Thomson Works.

[2] Wellman, "Iron Age," December 10, 1896.

[3] Richards, "Notes on Iron," p. 95.

[4] "Steel," p 354.

[5] "Iron Age," July 14, 1901.

[6] "Oesterreichisches Jahrbuch," 1901, p. 449.

[7] "Iron Age," December 10, 1896.

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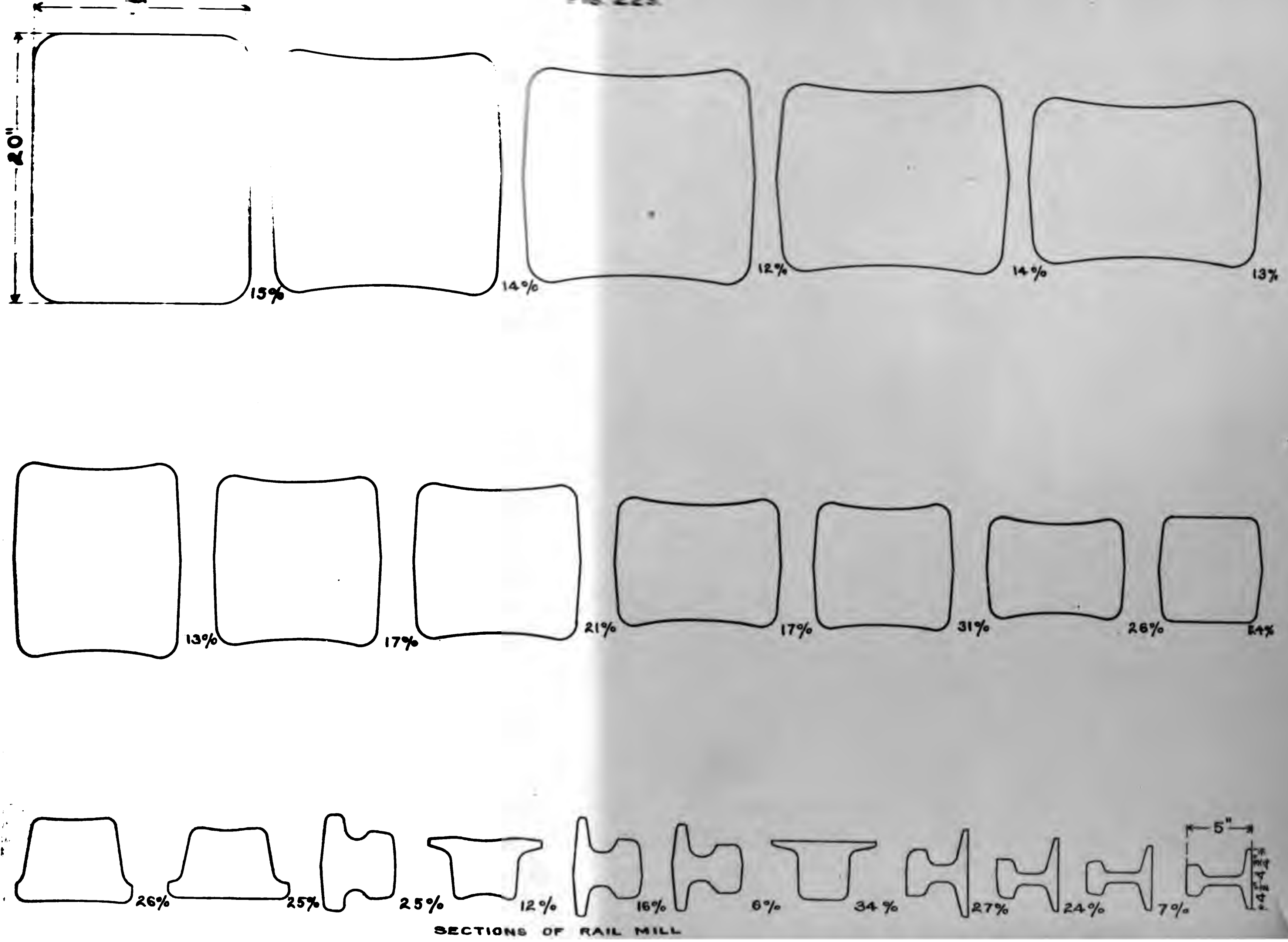
FIG. 225.



SECTIONS OF RAIL MILL

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SOUTH CHICAGO WORKS.

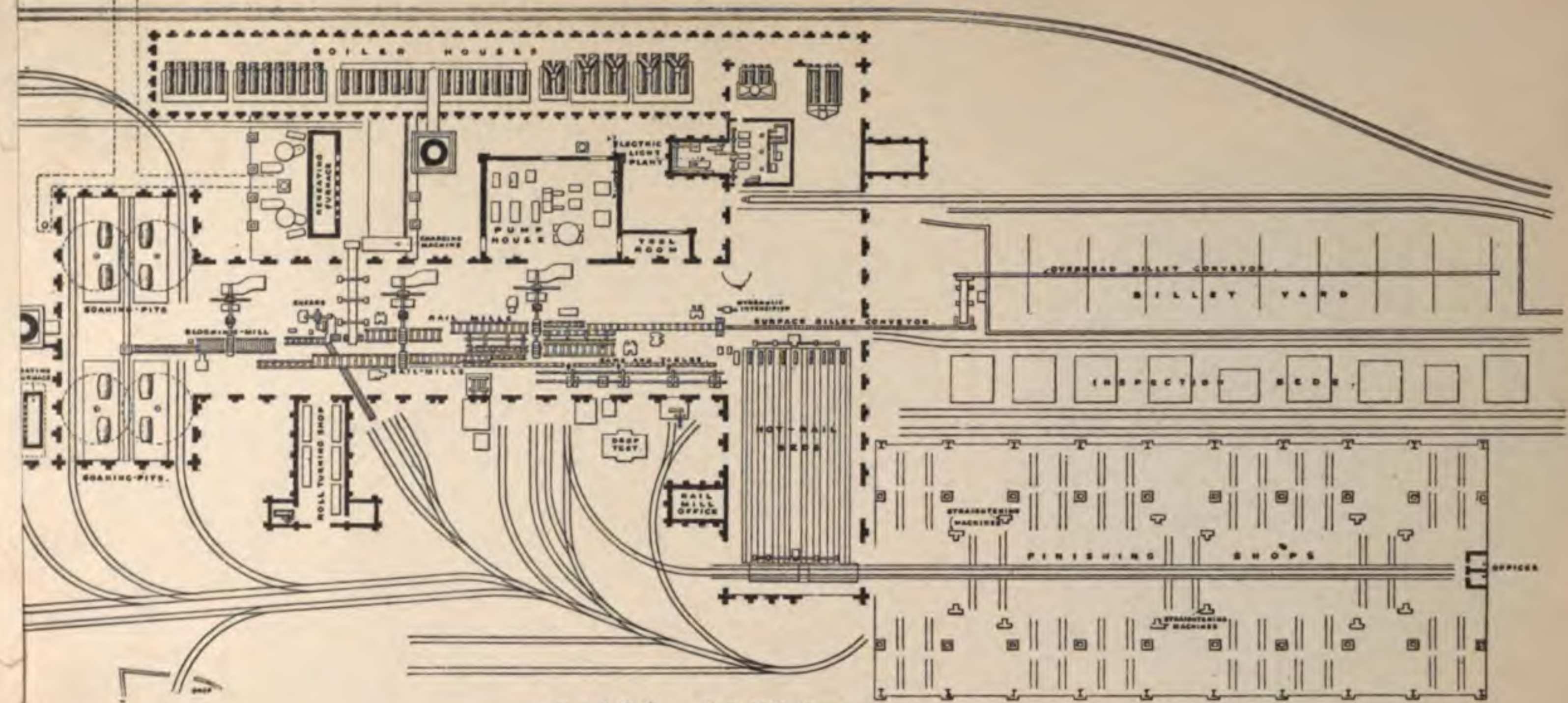
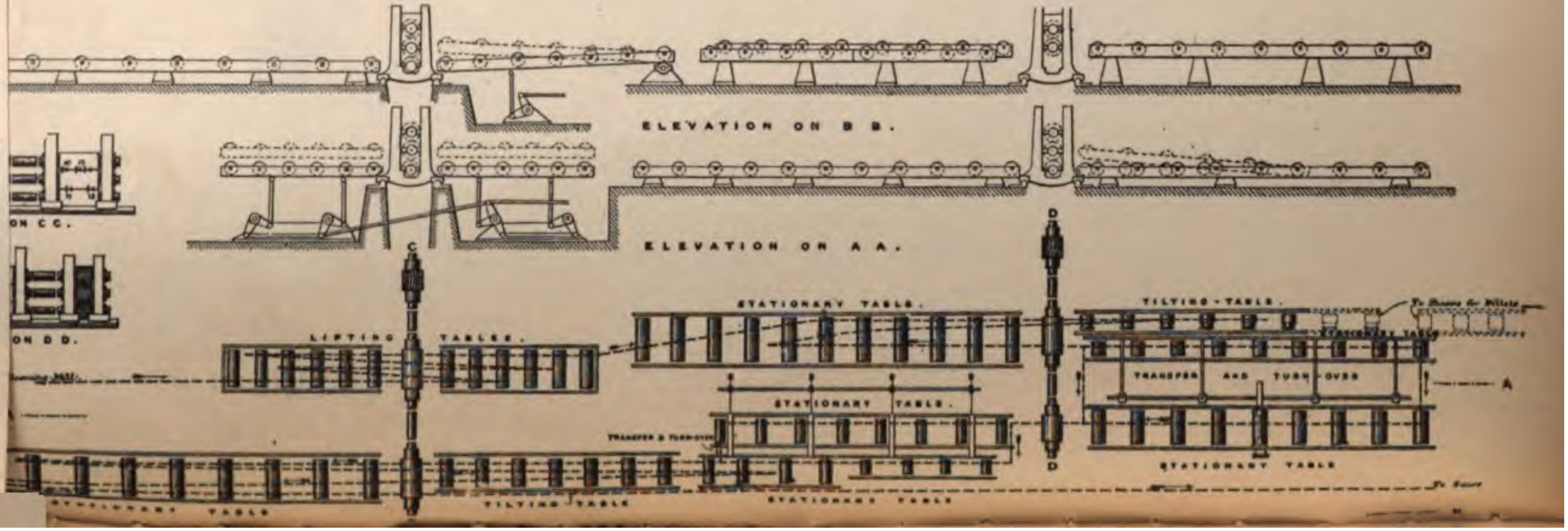


Fig. 223. Rail Mill Plan.



(Figures 213–216), arranged to be heated with producer gas or crude petroleum. There are eight furnaces, each holding eight ingots. From the furnaces the ingots go to the blooming mill and, *without reheating*,[1] to the rail mill. In the blooming mill they are rolled in nine passes into blooms, 7½ inches by 5 inches, which are cut[2] long enough to make three rails each, sometimes four, each ingot making two blooms. Figure 224 shows in plan and elevations the course taken by the ingot until finished to a rail. There are two rail trains, *C* and *D*, both 27 inches in diameter. The first train, *C*, consists of two stands and carries the first roughing rolls and finishing rolls. The engine is a Porter-Allen (54-inch cylinder, 66-inch stroke, eighty-five revolutions per minute). The second train, *D*, carries the second and third roughing rolls. The engine is also a Porter-Allen (44-inch cylinder, 66-inch stroke, eighty-five revolutions per minute). The bloom coming from the blooming mill passes through the different rolls, as shown by the dotted lines. The first roughing rolls are three-high and have lifting tables. The bloom is passed by means of the lifting tables through the first and second passes, when the table is lowered; the bloom is then automatically carried over to the third pass, and makes the third and fourth passes in the same way as the first and second. When the table is lowered, the bloom is turned 90° and then makes its fifth and last pass in the first roughing rolls. On leaving these, it is turned again 90° and is carried by rollers to the stationary table of the second roughing rolls *D*, and, making its sixth pass, comes on to the tilting table. As it leaves this pass it is again turned 90° by guides and by the shape of the grooved rollers in the tilting table. This table is lifted and the piece passes through the seventh pass, drops on the stationary table, and again enters the rolls. In the eighth pass, it is lifted from the tables by a transfer and carried over to the third set of rolls, in which one pass (the ninth) is made. From the stationary table it is transferred and turned 90°, dropping on the live rollers, which carry it to the finishing rollers. After four passes (the tenth to thirteenth), the finished rail passes over a stationary table to the hot saws, from which the single rails, after passing through the cambering machine, are delivered to the hot bed, etc. The capacity of the mill is great: 1,025 tons rails in twelve hours, 1,829 tons in twenty-four hours. In May, 1893, the production of rails was 38,093 tons, the full capacity being 50,000 tons. The greatest record is 1,442 tons in twelve hours.[3]

Figure 225 gives the sections from ingot down to finished rail of the blooming and rail mill of the Maryland Steel Company, Sparrow's Point, Maryland. The ingot, 26 x 16½ inches, is reduced in the blooming mill by ten passes to blooms 7¾ x 7¾ inches; a bloom goes straight, *without reheating*, to the rail mill, whence it issues after eleven passes as a finished rail, 5 x 5 inches. Photomicrographs are given in "Iron Age," December 26, 1901, p. 4.

Rail Mill of the Edgar Thomson Steel Works, Carnegie Steel Company, Pittsburgh.[4] The blooming mill is a three-high (Fritz) mill with fixed rolls 40 inches in diameter, 7 feet 4 inches long, drawn by an engine with 44-inch cylinder and 66-inch stroke. The ingot is 16¾ x 19 inches at the butt. Figures 226 and 227[5] show the plan of the works and the course

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[1] "Iron Age," December 26, 1901, p. 5.

[2] Aiken, Blooming Shear: "Iron Age," 1890, XLVI, p. 132; Daelen, "Stahl und Eisen," 1889, p. 23.

[3] "Engineering and Mining Journal," April 28, 1900.

[4] "Iron Age," December 10, 1896; "Cassier's Magazine," 1900, XVII, p. 272.

[5] "Iron Age," December 10, 1896.

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EDGAR THOMPSON WORKS.

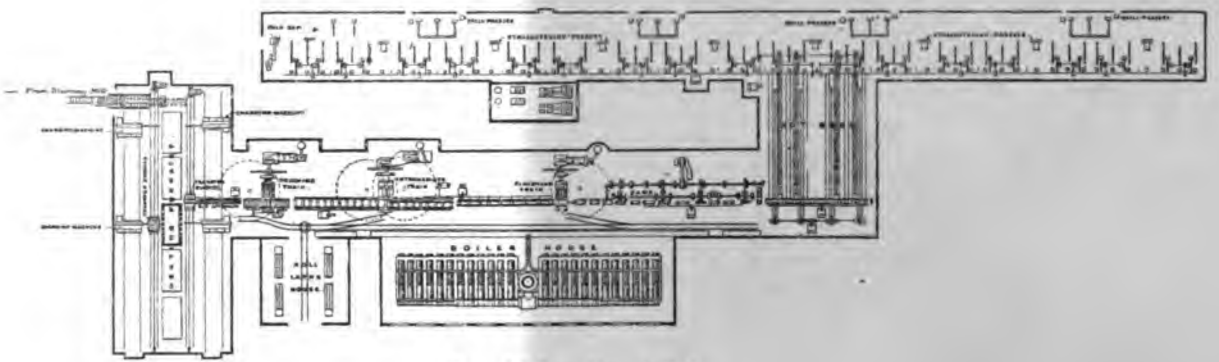

Fig. 226. Plan of Works.

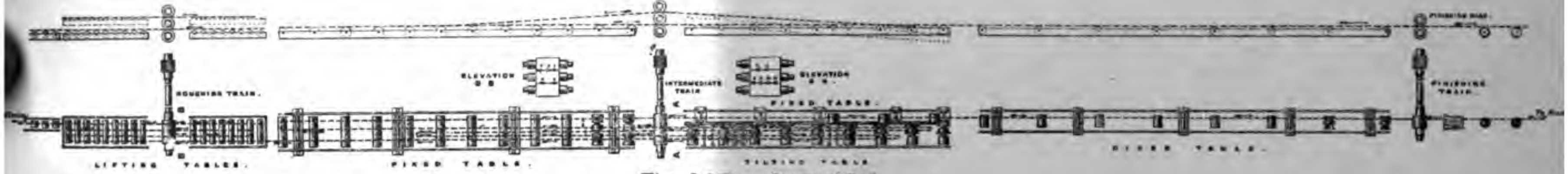

Fig. 227. Course of Rail.

of the rail. The 8 x 8-inch bloom from the blooming mill (Figure 226), after having been sheared, does ***not go directly*** into the rail mill, as at South Chicago, ***but first*** into a reheating furnace.[1] It is delivered by driving rolls from the shears to a transfer buggy[2] back of a row of reheating furnaces, which by means of an underground trolley may be moved in front of the doors of any of the furnaces. The bloom is pushed from the buggy by means of a Wellman charging machine[3] into the furnace. In front of the heating furnaces is a second transfer buggy onto which the hot bloom is drawn by another charging machine, which seizes it in the furnace.[4] From the transfer buggy the heated bloom goes onto the driven rolls of the rail-train approach. The rail train is divided into three sets of rolls $25\frac{1}{2}$ inches in diameter. The roughing train is a three-high mill drawn by an Allis engine, with 46-inch cylinder and 60-inch stroke, making eighty-five revolutions per minute. The intermediate train, 77 feet from the first train, also a three-high mill, is driven by a Porter-Allen engine, with 54-inch cylinder, 66-inch stroke, running at ninety revolutions per minute. The finishing train, 120 feet from the intermediate, is a two-high mill drawn by an Allis engine, with 30-inch cylinder, 48-inch stroke, making seventy revolutions per minute. The course of the bloom is shown in Figure 227. It makes five passes in the roughing train. Before the third and fourth pass, the piece is turned 90°. After the fifth pass, it goes to the intermediate train and is turned 90° before entering the sixth pass, and again before the seventh pass; after the tenth pass, the rail travels from the fixed table to the finishing train with one pass, when it is delivered to the hot saws (cutting the rail into 30-foot lengths), to the cambering machine, and to the hotbed (Figure 226), sets of skids made of rails. From the hotbed the rails are pushed[5] to short-driven rollers, distributing them to the straightening presses[6] to be first straightened, then drilled and inspected, when they are ready to be loaded into cars, which is done in part mechanically. So far the best month's work (October, 1894) was 36,390 tons rails; the best twenty-four hours' work 1,945 tons. This general outline has been changed by the introduction of the Kennedy-Morrison modification (see below) of the finishing temperature of the rail.

A Hot Saw[7] (Figure 228[8]) is a toothed circular disk of wrought iron or steel stiffened by cast-iron plates. The diameter of the saw varies from 30 to 60 inches, the thickness $\frac{1}{8}$ to $\frac{1}{4}$ inch, the length of the teeth from $\frac{9}{32}$ to $\frac{25}{32}$ inch, the distance of the teeth $\frac{9}{16}$ to $\frac{13}{16}$ inch, the number of revolutions per minute 800 to 2,000, giving a circumferential velocity of 200 to 260 feet per second.[9] Sometimes a plain iron or soft steel disk, 42 inches in diameter and $\frac{3}{16}$ inch thick, without teeth, with a circumferential velocity of 15,000 feet, is used, which will saw a hot rail through in one minute; the steel is, so to say, ground off and not cut off. A cold saw has

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[1] Drawing of Wellman Reheating Furnace with Regeneration Chambers: "Iron Age," April 28, 1892.

[2] Illustration: "Cassier's Magazine," 1899, XVII, p. 271; "Iron Age," September 18, 1902; "Scientific American," December 12, 1903.

[3] "Transactions American Institute of Mining Engineers," XIX, p. 313.

[4] Another form of apparatus for handling ingots is that by Kennedy-Porter, "Iron Age," December 4, 1890.

[5] Hotbed Conveyor: "Iron Age," March 10, 1904.

[6] "Iron Age," February 27, 1890.

[7] Flying Shears: Edwards, "Iron Age," June 5, 1902; Daelen, "Stahl und Eisen," 1903, p. 451; "Iron Trade Review," March 28, 1901.

[8] Richards, "Notes on Iron," p. 96.

[9] Ledebur, "Manual," 4th ed., p. 1132.

a smaller diameter than a hot saw; the ratio of thickness and diameter is greater; the teeth are shorter and placed more closely to one another; the velocity often does exceed 20 inches per second.

Martin[1] shows by interesting photomicrographs the superficial changes in structure a rail undergoes when cut with a hot saw and a cold saw.

A Cambering Rail is shown in Figure 222.[2] The rail leaves the finishing rolls straight; the head, being thicker than the flange, remains hotter and in cooling will contract more than the flange. The consequence is that the rail would become warped, as shown in *G*. In order to counteract this, the straight rail *E*, coming from the hot saws, is bent cambered in the opposite way, *F*, by rolls, and becomes approximately straight upon cooling on the hot-bed. *A* = pair of cambering rolls, *B* = cambering rollers, *C* = saw table with live rollers, *D* = gears, *E* = straight hot rails, *F* = cambered rail.

Finishing Temperature of Rails.[3] If carbon steel[4] is brought to a high temperature and allowed to cool slowly, crystals form; these crystals are the larger the higher the temperature and the slower the cooling; and large crystals (coarse structure) diminish the ductility, while small crystals (fine structure) increase it. If the steel is worked while cooling down from high temperature (hot work), crystallization is prevented. If the work is stopped above the critical range (say 700° C.), crystallization sets in and continues until the critical is reached; below this range no further crystals form. If working is continued below the range (cold work), the existing crystals are distorted and the steel becomes brittle. The ductility can be restored only by annealing.

In rolling steel blooms to rails the bloom ought not to be too hot at the start, and work ought to be stopped at as low a temperature as possible above the critical range, as the lower the finishing temperature the finer will be the crystals and consequently the more ductile the rail. Rolling *direct* requires heavier machinery and harder work than rolling after reheating. An ingot rolled direct, arrives as a bloom at the rail rolls at a temperature of about 850° C., while if reheated it will be delivered at 1,000° C. or more.[5] The South Chicago and Sparrow's Point Works roll direct; the Edgar Thomson Works roll with reheating. In order to overcome the disadvantage of coarser crystallization caused by the higher finishing temperature of the Edgar Thomson Works, and at the same time to retain the less powerful machinery, Kennedy-Morrison[6] have introduced a modification of the usual mode of procedure in that they hold the rail for a few minutes on a cooling table before delivering it to the last pass. While the photomicrographs published show the structure of the rails to have been improved, and

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[1] "Metallographist," 1902, V, p. 245.

[2] Richards, "Notes on Iron," p. 96.

[3] "Transactions American Institute of Mining Engineers," XXII, p. 540; XXIII, p. 633, Sauveur; XXXIV, Job, February, 1904, Hunt. "Iron Age," December 20, 1900, Kennedy-Morrison; December 26, 1901, Martin. "Iron Trade Review," December 20, 1900, Kennedy-Morrison; July 24, 1902, Job. "Proceedings American Society for Testing Materials," 1902, II, p. 75, Martin; p. 79, Sauveur; 1903, III, p. 278, Sauveur-Whiting. "Metallographist," V, p. 177, Job, p. 191, Martin; p. 197, Sauveur; VI, p. 111, Dudley. "American Manufacturer," July 2, 9, 16, 1903, Dudley. "Journal Iron and Steel Institute," 1901, II, p. 52, Risdale; 1903, II, p. 36, Campbell; p. 333, Lloyd. "Revue Universelle des Mines," 1901, LIV, p. 120, Pasquier. "Iron and Steel Magazine," I, p. 203, Sauveur-Whiting. See also these Notes, Chapter V, J.: Hardening, Annealing, Tempering.

[4] Sauveur, "American Society Testing Materials," 1902, II, p. 197; "Metallographist," 1902, V, p. 197.

[5] Martin, "Iron Age," December 26, 1901; "Metallographist," V, p. 191.

[6] "Iron Age" and "Iron Trade Review," December 20, 1890.

a smaller diameter than a hot saw; the ratio of thickness and diameter is greater; the teeth are shorter and placed more closely to one another; the velocity often does exceed 20 inches per second.

Martin[1] shows by interesting photomicrographs the superficial changes in structure a rail undergoes when cut with a hot saw and a cold saw.

A Cambering Roll is shown in Figure 229.[2] The rail leaves the finishing rolls straight; the head, being thicker than the flange, remains hotter and in cooling will contract more than the flange. The consequence is that the rail would become warped, as shown in G. In order to counteract this, the straight rail E, coming from the hot saws, is bent (cambered) in the opposite way, F, by rolls, and becomes approximately straight upon cooling on the hot-bed. $A =$ pair of cambering rolls, $B =$ cambering rollers, $C =$ saw table with live rollers, $D =$ gears, $E =$ straight hot rails, $F =$ cambered rail.

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In rolling steel blooms to rails the bloom ought not to be too hot at the start, and work ought to be stopped at as low a temperature as possible above the critical range, as the lower the finishing temperature the finer will be the crystals, and consequently the more ductile the rail. Rolling *direct* requires heavier machinery and harder work than rolling after reheating. An ingot rolled direct, arrives as a bloom at the rail rolls at a temperature of about 850° C., while if reheated it will be delivered at 1,000° C. or more.[5] The South Chicago and Sparrow's Point Works roll direct; the Edgar Thomson Works roll with reheating. In order to overcome the disadvantage of coarser crystallization caused by the higher finishing temperature of the Edgar Thomson Works, and at the same time to retain the less powerful machinery, Kennedy-Morrison[6] have introduced a modification of the usual mode of procedure in that they hold the rail for a few minutes on a cooling table before delivering it to the last pass. While the photomicrographs published show the structure of the rails to have been improved, and

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[1] "Metallographist," 1902, V, p. 245.

[2] Richards, "Notes on Iron," p. 96.

[3] "Transactions American Institute of Mining Engineers," XXII, p. 546; XXIII, p. 653, Sauveur; XXXIV, Job; February, 1904, Hunt. "Iron Age," December 20, 1900, Kennedy-Morrison; December 26, 1901, Martin. "Iron Trade Review," December 20, 1900, Kennedy-Morrison; July 24, 1902, Job. "Proceedings American Society for Testing Materials," 1902, II, p. 75, Martin; p. 79, Sauveur; 1903, III, p. 278, Sauveur-Whiting. "Metallographist," V, p. 177, Job; p. 191, Martin; p. 197, Sauveur; VI, p. 111, Dudley. "American Manufacturer," July 2, 9, 16, 1903, Dudley. "Journal Iron and Steel Institute," 1901, II, p. 52, Risdale; 1903, II, p. 36, Campbell; p. 353, Lloyd. "Revue Universelle des Mines," 1901, LIV, p. 120, Pasquier. "Iron and Steel Magazine," I, p. 203, Sauveur-Whiting. See also these Notes, Chapter V, J.: Hardening, Annealing, Tempering.

[4] Sauveur, "American Society Testing Materials," 1902, II, p. 197; "Metallographist," 1902, V, p. 197.

[5] Martin, "Iron Age," December 26, 1901; "Metallographist," V, p. 191.

[6] "Iron Age" and "Iron Trade Review," December 20, 1890.

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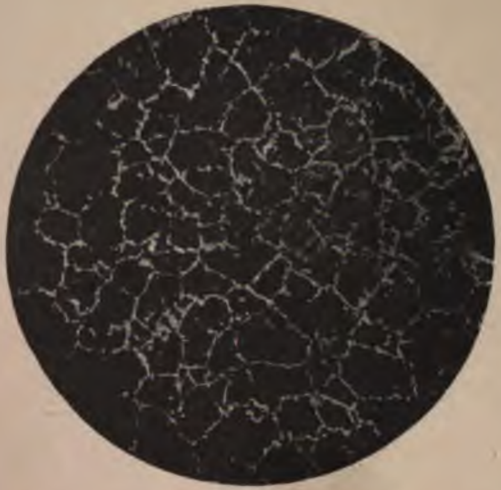
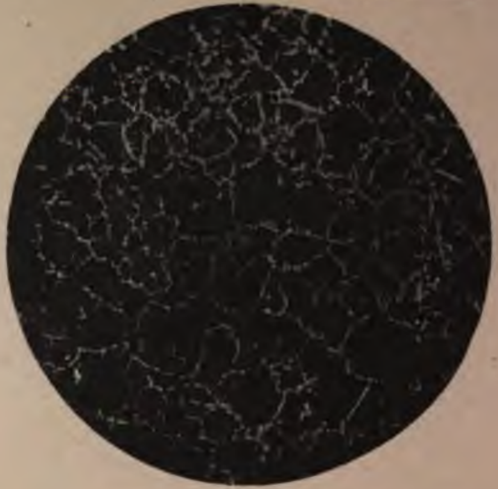
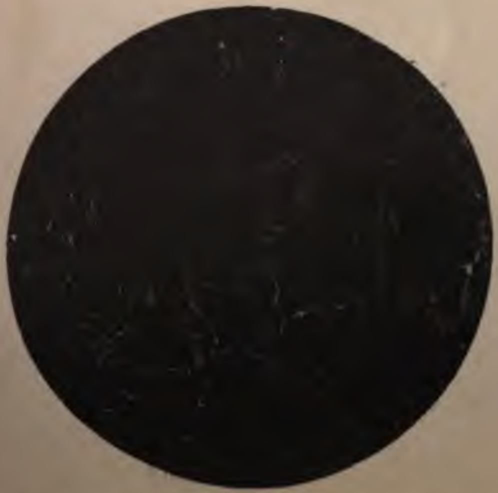
MARTIN ON RAIL TEMPERATURES.



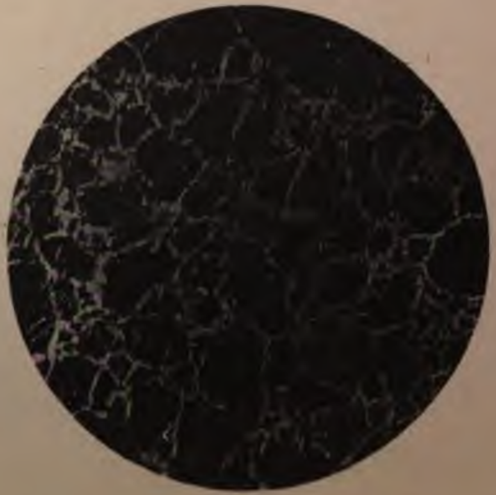
From center of head of 100-lb. rail rolled direct at low temperature throughout the rail rolls. Shrinkage: 7 in. in 33 feet. Magnified 45 diam.



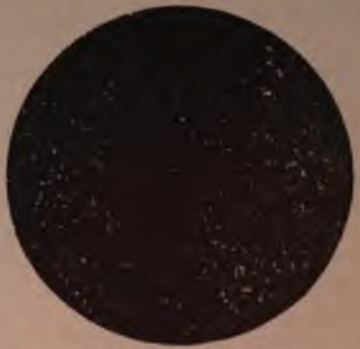
From center of head of 100-lb. rail rolled direct, but at a higher temperature than Fig. 1, and held before finishing pass until cold enough to give 6½ in. shrinkage in 33 feet. Structure larger than in Fig. 1. Magnified 45 diam.



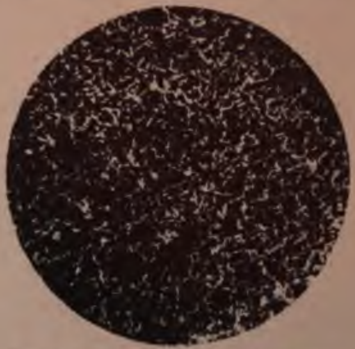
From center of head of 100-lb. rail rolled by usual process from reheated bloom. Shrinkage: 7⅛ in. in 33 feet. Magnified 45 diam.



From center of head of 100-lb. rail rolled by Morrison-Kennedy process from reheated bloom, held before finishing pass. Shrinkage, 6 in. in 33 feet. Magnified 45 diam.



SECTION FROM HEAD OF SEVENTY-FIVE POUND RAIL



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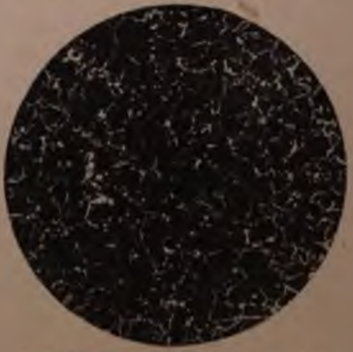

SECTION FROM HEAD OF HUNDRED POUND RAIL

while continued use of the process speaks in favor of it, it is justly argued by Sauveur[1] that the total crystallizing has not been shortened, but only divided in two. Sauveur states that the real improvement in this case lies in rolling at a temperature most desirable for saving in power and increasing of speed, and then annealing the finished rail at a temperature slightly above 700° C.

Photomicrographs by Martin (Figures 230–234)[2] show the results of the different procedures, and require no further comment, excepting as to the shrinkage, inches in feet, given which indicates the temperature at which the rail was finished. Martin (*l.c.*) found that a shrinkage of from $5\frac{5}{8}$ to $6\frac{1}{4}$ inches in 33 feet corresponded to a range of temperature of from 780° C. to 760° C. Sauveur-Whiting[3] constructed a thermomagnetic selector which is to separate magnetic rails, finished at the correct temperature, from non-magnetic rails finished at too high a temperature. Figures 235–237, photomicrographs also by Martin, show the effect of work on the microstructure of direct-rolls rails of different weights; the lighter the rail, the deeper did the work penetrate and the finer was the structure. But a thick piece cannot be finished at a uniform temperature, as there will always be differences in temperature between the outside and inside. If the outside is right, the inside will be too hot; if the inside is right, the outside will be too cool. This is clearly shown in the photomicrographs (Figures 238–239) of a steel rail.[4]

Specifications of Steel Rails: "Proceedings American Society for Testing Materials," 1903, III, p. 75.

References for Mills: Development of American Blooming Mills: "American Manufacturer," 1897, LXI, pp. 443, 476. Rail Mill, Ohio Steel Company, Youngstown, Ohio: "Iron Age," August 16, 1894. Rail Mill of Joliet: *op. cit.*, December 10, 1896. Mill for Rerolling Rails: *op. cit.*, March 20, 1903. Bessemer plant, Johnson Company, Lorain, Ohio: *op. cit.*, May 9, 1895. Bessemer plant, National Tube Works, McKeesport, Pennsylvania: *op. cit.*, November 16, 1893. Bessemer plant, Shenango Valley Steel Company, Newcastle, Pennsylvania: *op. cit.*, October 13, 1892. Bessemer plant, Lackawanna Steel Company, Buffalo, New York: *op. cit.*, January. 4, 1904.

B. THE BASIC BESSEMER PROCESS.

Literature: Wedding (translated by Phillips-Prochaska), "Basic Bessemer Process," New York, 1891; Thomas-Gilchrist, "Journal Iron and Steel Institute," 1879, I, p. 120; 1882, I, p. 192; "Stahl und Eisen," 1882, p. 294; Ehrenwerth, "Studien über den Thomas-Gilchrist Process," Vienna, 1881; reprint from "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1880 and 1881; Sjörgen, "Oesterreichisches Jahrbuch," 1899, XLVII, p. 151.

SECTION 59. **In General.** The feature which principally distinguishes the basic from the acid Bessemer process is that the operation is carried on in a basic-lined vessel in the presence of burnt lime. The Si of the pig, when oxidized to SiO_2 combining with an excess of

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[1] "Metallographist," V, p. 200.

[2] "Proceedings American Society for Testing Materials," 1902, II, p. 78.

[3] *Op. cit.*, 1903, III, p. 278; "Iron and Steel Magazine," I, p. 503.

[4] Sauveur, "Transactions American Institute of Mining Engineers," XXII, p. 542, Plate IV.

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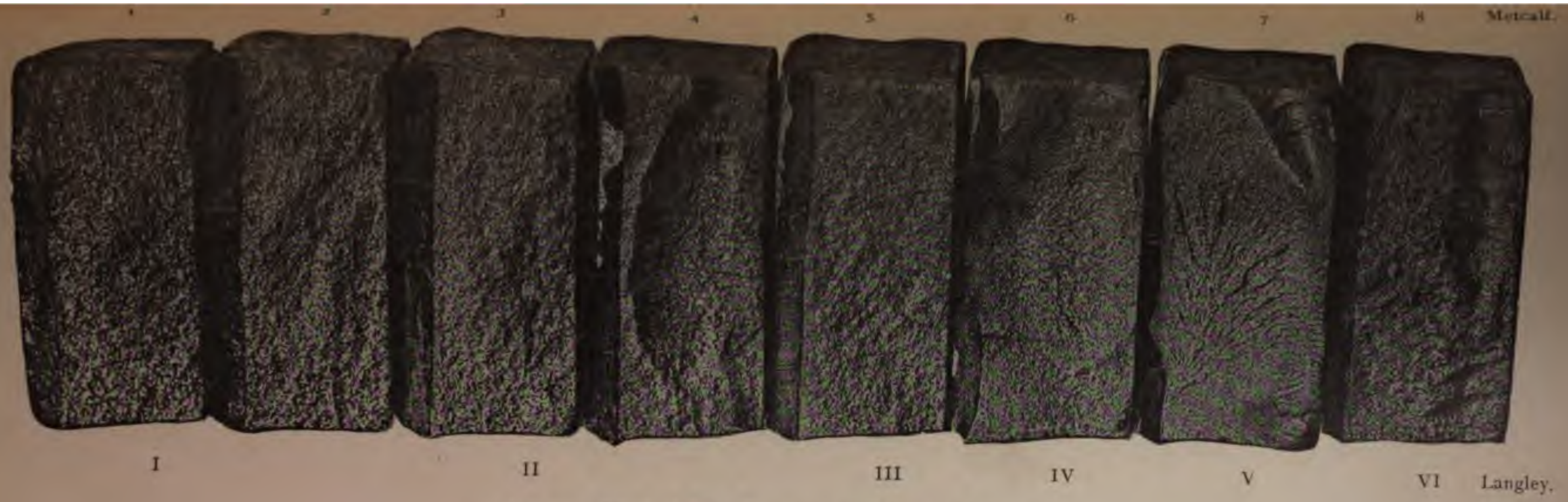

Fig. 314. Effect of Heat Treatment on Fracture of Steel.

base, forms a basic slag which permits the removal of P, and the presence of non-combined CaO makes possible the removal of at least part of the S.[1] The heat necessary for the process is furnished mainly by the oxidation of P to P_2O_5 (see p. 134). The process was carried out at Pottstown, Pennsylvania; the works were closed in 1893. It is not used in this country at present, as it is more expensive than the acid process, and as there is still an abundance of acid Bessemer ore. It is also stated that the elimination of P is not under absolute control,[2] the end product showing considerable variation. The process is developed mainly in Germany.

The General Arrangement of a basic plant resembles very much that of an acid plant; in addition to what is necessary for the latter, there have to be provided apparatus for preparing the basic lining and the burnt lime, and for charging the lime into the converter. For drawings of the Pottstown, Pennsylvania, plant, see "Iron Age," May 28, 1891; "Transactions American Institute of Mining Engineers," XXI, p. 743; of the Troy, New York, plant: "Iron Age," January 16, 1896. Older plants: "Stahl und Eisen," 1890, pp. 939, 1041; 1891, p. 730; 1893, p. 101; "Annales des Mines de Belgique," 1902, VII, p. 279. Newer plants: see arrangement of acid plants, p. 134.

The Raw Materials are basic pig iron, spiegeleisen, and ferromanganese. Basic pig contains[3] Si, 0.2–0.5 per cent.; P, 1.9–2.7 per cent.[4]; Mn, 1–2 per cent.; C, 3–3.5 per cent.; S is usually higher (under 0.12 per cent.) than in the acid pig.[5] P must be high to furnish the necessary heat, Si low to keep the slag basic and small in quantity, otherwise P would not be slagged and the converter lining would be corroded. The pig iron, as in the acid process, is melted in the cupola. As long as the melted pig iron retains 0.75 to 1.00 per cent. Mn, the S in the resulting steel will be below 0.05 per cent.[6] Direct metal is also used in connection with mixers[7] having a capacity of 100–150 tons.[8] Spiegeleisen and ferromanganese are the same as in the acid process.[9]

The Converter has the same general exterior form as with the acid process.[10] The lining and bottom are of burnt dolomite or limestone.

CaO and MgO are infusible alone or when mixed, MgO being more refractory than CaO. Small admixtures of 1–2 per cent. each of SiO_2, Al_2O_3, and Fe_2O_3 do no harm, in fact may be of advantage as they assist fritting when the dolomite is being calcined, and thus reduce the slacking.

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[1] Snelus, "Journal Iron and Steel Institute," 1879, I, p. 135.

[2] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 185.

[3] Grassman, "Stahl und Eisen," 1896, p. 57.

[4] Howe, "Engineering and Mining Journal," November 22, 1890.

[5] See also Kintzlé, "Stahl und Eisen," 1897, p. 381; Thompson, "Iron Age," January 18, 1894.

[6] Thompson, "Iron Age," January 18, 1894.

[7] See Figure 136, Hoerde Mixer.

[8] Kintzlé, "Stahl und Eisen," 1897, p. 381.

[9] Analyses, see p. 94.

[10] Illustration of Pottstown: "Transactions American Institute of Mining Engineers," XXI, p. 751.

ANALYSES OF DOLOMITE.

	1	3	5	7	9	11	13	15
CaO	30.57	31.49	29.86	31.36	30.09	35.77	42.83	45.70
MgO	20.99	20.53	20.17	19.28	19.12	15.82	8.38	4.91
FeO	. . .	. . .	. . .	. . .	1.40	0.56	Trace	. . .
MnO	. . .	. . .	. . .	. . .	. . .	0.04	. . .	. . .
CO_2	47.10	47.32	45.64	45.86	45.59	45.89	42.85	41.31
P_2O_5	. . .	. . .	. . .	. . .	0.03	. . .	Trace[9]	. . .
SiO_2	. . .	. . .	4.34	2.00	3.04	. . .	0.44	1.68
Al_2O_3	. . .	. . .	. . .	1.50	0.73	1.05	0.12	1.88
Fe_2O_3	0.86[1]	0.51[3]	. . .	. . .	. . .	. . .	0.66	. . .
Mn_2O_3	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .
H_2O	0.48	0.15	. . .	. . .	. . .	. . .	4.51[8]	4.36

[1] Including 0.44 per cent. of clay and sand.
[3] Including 0.13 per cent. of clay and sand.
[8] Including organic matter.
[9] Also a trace of alkalies.

REFERENCE. — Wedding (Phillips-Prochaska), "Basic Bessemer Process," p. 40.

Dolomite is calcined[1] in a cupola similar to that used in melting pig iron excepting that near the bottom there are four to six drawing doors to remove the calcined material. A cupola is used, as it is essential to calcine at a white heat. Such a cupola has, *e.g.*, the following dimensions: inner diameter, 6 feet; height to tuyères, 4 feet; to feed door, 16 feet; there are eight tuyères 4 inches in diameter; the lower 5 feet are lined with dolomite, the rest with fire brick followed by red brick. In starting, the cupola is half filled with coke, then follow alternate layers of dolomite (first size) and coke (6–8 per cent.). The capacity for twenty-four hours is 14 tons calcined material. This is crushed in a dry pan to pass a ¼-inch sieve and mixed with 7–10 per cent. dehydrated tar in a steam-heated pan by hand or machinery to a mass, which is plastic while warm and hardens upon cooling. The mixture is either rammed with hot irons into the vessel around a centre or it is first formed into bricks, which are kiln-burned and then laid in a mortar, which is a mixture of burned dolomite dust and tar. Both pin bottoms and fire-brick tuyères are used. The acid brick tuyères are quickly eaten away by the basic slag, but can be readily replaced. Magnesite tuyères have not been used so far. At the Pottstown (Pennsylvania) Basic Bessemer Plant,[2] the dolomite was prepared by passing through a crusher, pulverizing in a dry pan, mixing with 2 per cent. basic cinder, briquetting, kiln-burning (5,000–6,000 bricks in thirty-six hours; shrinkage in size and weight, 50 per cent.), crushing, screening, and mixing with 10 per cent. tar free from water. When the lining of the vessel is heated, the tar is converted into coke, which serves as a bond. By the time the coke is burned out,

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[1] Cost: "Iron Age," May 3, 1894, p. 846, col. 1. Plant: "Stahl und Eisen," 1902, p. 1201; Baker, "Iron Age," April 21, 1904.

[2] "Iron Age," May 28, 1891; "Transactions American Institute of Mining Engineers," XXI, p. 745.

the lining has become partially fused and come to a "set." The thickness of the lining at the bottom varies from 8 to 26 inches, at the nose from 6 to 18 inches. The lining, however, unfortunately continues to shrink, even when calcined at a white heat. This causes the formation of small cracks and shortens the life. A lining lasts 100–120 heats.[1] As it cannot be repaired as quickly as in the acid process, a basic plant has three converters as against the two of the acid process. A bottom usually lasts twenty heats; if rammed with the Versen machine,[2] it lasts twenty-five to forty heats.[3] There are required per ton ingot steel 60–70 pounds dolomite and 5–6 pounds tar.[4]

Ladles, Cranes, Molds, Blowing Engines, etc., are of the same general character as in the acid process.

SECTION 60. **Chemistry, Mode of Operating, and Results.** The changes that basic pig iron undergo in the converter are shown by the two following examples of Ruhrort[5] and ~~of~~ Hoerde.[6]

RUHRORT.

	Time from beginning.	METAL.					SLAG.						
		Si	C	P	S	Mn	SiO_2	CaO	P_2O_5	FeO	Fe_2O_3	MnO	MgO
Fore blow.	Pig iron[1]	1.22	3.21	2.183	0.080	1.03	. . .	. . .	. . .	. . .	. . .	. . .	. . .
	2 minutes 46 seconds	0.72	3.30	2.148	0.047	0.71	41.15	41.27	0.84	2.40	. . .	9.03	4.13
	5 minutes 21 seconds	0.15	3.12	2.224	0.051	0.50	36.30	39.50	3.12	3.97	0.46	11.02	3.39
	8 minutes 5 seconds	0.007	2.47	2.157	0.049	0.18	34.41	42.80	2.99	3.60	0.13	10.72	3.35
	10 minutes 45 seconds	0.012	1.49	2.096	0.051	0.16	31.94	43.12	4.02	4.23	0.74	9.94	4.01
	13 minutes 28 seconds	0.005	0.75	2.053	0.051	0.14	16.64	44.37	7.15	8.42	4.95	8.51	7.34
	15 minutes 13 seconds	0.008	0.05	1.910	0.055	0.01	14.65	46.63	11.60	7.15	3.84	7.39	6.34
After blow.	19 minutes 14 seconds	0.005	0.02	0.230	0.060	0.01	12.94	47.76	18.83	5.84	3.74	4.25	6.00
	19 minutes 31 seconds	0.005	0.02	0.139	0.055	. . .	12.20	48.59	18.66	6.79	2.80	4.01	6.26
	19 minutes 49 seconds	0.004	. . .	0.087	0.056	. . .	11.71	48.19	18.15	7.19	2.78	4.05	6.38
	Rail steel	0.01	0.26	0.145	0.045	0.48	12.77	47.87	16.92	5.94	2.87	4.80	6.75

[1] 12–15 per cent. lime added.

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[1] "Stahl und Eisen," 1885, p. 383; "Iron Age," September 4, 1890.

[2] "Stahl und Eisen," 1892, p. 1089; 1893, p. 919; "Zeitschrift des Vereins deutscher Ingenieure," 1902, p. 1543; 1903, p. 421.

[3] Kintzlé, "Stahl und Eisen," 1897, p. 381; "Berg- und Hüttenmännische Zeitung," 1897, p. 189.

[4] Kintzlé, "Stahl und Eisen," 1897, p. 381.

[5] Wedding (Phillips-Prochaska), "The Basic Bessemer Process," pp. 136, 137, 143.

[6] Wedding, "Transactions American Institute of Mining Engineers," XIX, p. 366.

HOERDE.

	Time from beginning.	METAL.					SLAG.						GAS.[1]				
		Si	C	P	S	Mn	SiO_2	CaO	P_2O_5	FeO	MnO	MgO	CO_2	O	CO	CH_4	N
Fore blow . . .	Pig iron	0.58	3.60	2.75	0.079	1.57	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .
	About 3 minutes	0.28	2.81	2.57	0.079	2.50	9.20	76.10	2.94	0.55	8.87	4.86	3.5	8.1	2.0	0.9	85.0
	About 6 minutes	0.07	2.02	2.08	0.073	0.30	9.50	71.40	6.90	0.73	9.70	5.33	3.0	3.0	10.6	1.0	81.4
	About 9 minutes	0.07	1.33	2.25	0.074	0.34	9.30	66.17	7.32	2.30	8.42	6.47	3.0	0.3	28.3	1.6	66.6
	About 12 minutes	0.06	0.71	1.55	0.079	0.26	10.28	50.71	15.87	7.13	9.45	6.90	1.8	0.2	29.8	1.8	65.0
After blow . . .	About 15 minutes	0.02	0.105	0.061	0.054	0.21	6.99	46.84	24.73	11.98	5.40	4.09	1.2	0.8	1.6	0.9	95.6
	Steel	0.02	0.136	0.084	0.046	0.55	4.79	42.05	16.33	26.03	4.62	6.83	. . .	. . .	. . .	. . .	. . .

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[1] Samples of metal and slag were taken after the corresponding gas sample.

The changes in the charge at Ruhrort are graphically represented in Figure 240.

Supposing the converter to be turned horizontally and the blast shut off. The pig metal is run into the converter and is followed by 12–15 per cent. of burnt lime, introduced through a chute or some convenient device.

Sometimes only two-thirds of the total lime is charged at first, and the rest toward the end of the blow after the first slag has been drawn off. According to Hilgenstock,[1] at least enough CaO must be added to form $4CaO.P_2O_5$ and $CaO.SiO_2$. The first slag, rich in P and poor in Fe, is well suited as a fertilizer, while the second slag, poor in P and rich in Fe, goes back to the blast furnace.

The blast is now turned on and the converter turned up. The blow may be divided[2] into two periods, the *Fore Blow* and the *After Blow*. The *Fore Blow* is similar to that in the acid converter. First the Si will burn; when this has been eliminated, C begins to burn until it has been reduced to about 0.05 per cent.; Mn will be eliminated in part while the Si is being oxidized, the rest during the combustion of C. In the presence of much Mn in the slag, some may be reduced by the P[3] as follows: $4CaO + 5MnO + 2P = 4CaO.P_2O_5 + 5Mn$. Very little P is removed in this period. The Si will be more completely removed than in the acid converter, as its percentage is low at the start, as the basic CaO will readily combine with it, and as the heating and melting of the CaO, charged before blowing, absorbs so much heat that the critical temperature at which C burns is not reached before the Si is oxidized. With the burning out of the C the flame drops,[4] the fore blow ceases, and the after blow begins, in order to eliminate P and to some extent S. P is oxidized, the P_2O_5 combining with CaO to a phosphate is taken up by the basic slag. Any P oxidized before all the Si had been burned and combined with an excess of base would be reduced again, the SiO_2 uniting with the base combined with the P_2O_5, and the P_2O_5 being reduced by metallic iron. S is removed at the same time as P. If present in a large amount, the blow may have to be continued and iron burned after the P has been eliminated.[5] As there is no flame issuing from the nose of the converter during the after blow, the end of this cannot be recognized in the usual way. It is necessary to regulate[6] the amount of air blown into the furnace by the P present; the engine is allowed to make a certain number of revolutions, which makes the air cylinder furnish a corresponding amount of air. Formerly it was the common practice to turn down the converter, take a ladle sample, hammer the steel flat, break it, and judge the end of the process by the fracture. In many cases this has been found unnecessary.[7] A blow lasts twelve to eighteen minutes, two-thirds of the time being taken up by the fore blow. Kintzlé[8] states that with 12–15 ton converters the blow takes one minute for 1 ton of ingot. When the blow is finished, the slag is poured off and the metal recarburized by spiegeleisen or ferromanganese, or, if the percentage of Mn is to

[1] "Stahl und Eisen," 1886, p. 525; 1887, p. 557.

[2] See Thompson, "Iron Age," December 19, 25, 1895.

[3] Stead, "Journal Iron and Steel Institute," 1893, I, p. 63.

[4] Hartley-Rammage, Spectra Flames of Different Periods: "Journal Iron and Steel Institute," 1901, II, p. 197.

[5] Stead, "Journal Iron and Steel Institute," 1893, I, pp. 61 and 62.

[6] "Stahl und Eisen," 1896, pp. 50, 124, Brovot; p. 206, Risdale.

[7] Thompson, "Iron Age," December 19, 25, 1895.

[8] "Stahl und Eisen," 1897, p. 381.

be kept low, by finely crushed coke free from S (Darby recarburization). The ferromanganese or coke is usually charged into the ladle. As the steel contains more oxide than in the acid process, and the slag contains oxide of iron disseminated through it, a very strong reaction takes place, and the CO set free reduces some of the P of the slag (see table *ante*). Today this is mostly avoided by keeping the temperature of the metal low, by using an excess of lime, and by chilling the slag before pouring.[1]

The following two tables give a summary of the data of a twelve-hour day's run (twenty-three heats) at Pottstown,[2] and of the conditions of 102 heats (selected from a total of 1,084 made in three months) which with low lime and hot casting temperature gave a low (< 11 per cent.) conversion loss: —

Number of heats made	23
Per cent. pig in cupola mix	90.50
Per cent. scrap in cupola mix	9.50
Average per cent. P in mix	2.49
Average per cent. conversion loss	14.10
All temperatures	hot
Per cent. lime charged	15 and 16
Average per cent. P in steel	0.041
Highest per cent. P in steel	0.070
Number revolutions in fore blow	732–900
Number revolutions in after blow	475
Charge of iron, pounds	25,000

All slags	sticky
Time of blow, minutes	13–15
Number of heats in averages	102
Average per cent. conversion loss	9.16
Average per cent. lime charged	13.00
Number revolutions in fore blow	800–1,100
Average scrap addition, pounds	900
Per cent. P in pig iron	2.20–2.90
Times of blows, minutes	13–17
All temperatures	hot
Average per cent. P in steel	0.044
Average blast pressure, pounds	30

SECTION 61. **Evolution of Heat.** The following calculation[3] is based on 1,000 kilograms pig iron with Si, 0.50 per cent.; P, 1.5 per cent.; C, 4.0 per cent., and presupposes that 4 per cent. Fe is burned to useful purpose: —

	Calories.
Combustion of 5 kilos of Si develop	23,430
Combustion of 35 kilos of C develop	40,700
Combustion of 40 kilos of Fe develop	29,640
Combustion of 15 kilos P + 19.4 kilos O = 34.4 kilos P_2O_5 develop .	57,315
Total	151,085

It shows that the P contributes over one-third of total heat evolved, and that in the basic process (in spite of the low Si) 50 per cent. more heat is theoretically evolved than in the acid process.

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[1] See Hartshorne, "Transactions American Society of Civil Engineers," XXXIII, p. 384.

[2] Thompson, "Iron Age," December 19, 25, 1895, p. 1262.

[3] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 140.

The loss of iron in converting is (Kintzlé, *l. c.*) from 13 to 16 per cent. (*versus* 10 per cent. in the acid process). Grassmann[1] says that at first it amounted to 17 per cent. and has now been reduced to 11 per cent.

The *Duplex Process*[2] for converting pig iron running high in Si as well as in P consists in blowing in an acid converter to eliminate the Si and part of the C, and then transferring the metal to a basic open-hearth furnace to eliminate the rest of the C and the P. Kernohan[3] substituted for the converter an inclined oblong chamber (the reducer) having tuyères in the bottom. The hot metal while flowing down the reducer in from five to six minutes in a layer of about 3 inches thick is blown (blast pressure, 10 pounds) as in the converter, and leaving it is collected in a hot-metal ladle and conveyed to the open-hearth furnace. In the Lash Process,[4] the pig iron is blown in an acid converter and the highly oxidized metal transferred to a basic open-hearth furnace charged with melted pig iron and poured into it. The contact of the two metals causes a violent reaction to take place. Goldstein[5] proposes to carry on Bessemer and open-hearth work in a single tilting furnace.

The Electric Production of Steel is discussed in Section 84.

SECTION 62. **Products.** The products are ingot steel, slag, gases, fumes, and odds and ends.

(1) *Ingot Steel.* Basic Bessemer steel is used in Europe, especially Germany,[6] for the same purposes that acid Bessemer steel is used here. Objections have been made to its use by Campbell.[7] Hartshorne[8] and Thompson[9] defend it. Thompson, giving full details as to the properties of the steel produced in Pottstown in 1890–93, shows that it is as regular and uniform in its properties as acid Bessemer steel. Of course, basic steel will, as a rule, run lower in Si and P than acid steel.

(2) *The Slag.*[10] Some of the slag, especially the end slag, goes back to the blast furnace to increase the percentage of P of the pig. The first slag produced, when running over 16 per cent. P and 40 per cent. CaO, forms a valuable fertilizer without further treatment[11] if ground sufficiently fine. At Pottstown, Pennsylvania,[12] the slag with 21 per cent. P was ground in a ball mill, so that 75 per cent. passed through a 150-mesh sieve and 90 per cent. through a 100-mesh sieve.

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[1] "Stahl und Eisen," 1896, p. 57.

[2] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 337; Ledebur, "Manual," 4th ed., p. 1037.

[3] "Iron Age," January 31, February 7, 1901.

[4] "Iron Trade Review," June 9, 1904.

[5] "Stahl und Eisen," 1904, p. 341.

[6] Structural Steel for Bridges: "Stahl und Eisen," 1893, p. 581.

[7] "Transactions American Society of Civil Engineers," XXX; "Iron Age," April 11, 1895, p. 768.

[8] "Iron Age," *l. c.*

[9] *Op. cit.*, August 8, 1895.

[10] A. Wiesner, "Thomasschlacke und Natürliche Phosphate," Vienna, 1895; M. Passon, "Das Thomasmehl," Neudamm, 1901.

[11] "Stahl und Eisen," 1885, p. 593; 1886, pp. 59, 688; 1890, p. 310.

[12] Thompson, "Iron Age," January 7, 1897; Morris, "Transactions American Institute of Mining Engineers," XXI, p. 232.

SLAG ANALYSES.[1]

SiO_2	5.76	6.77	16.41	6.69	7.07	7.73	5.37	6.99	4.70
P_2O_5	19.19	16.92	11.75	17.75	22.50	21.90	20.96	24.73	16.33
Al_2O_3	1.43	1.68	1.58	0.95	0.89	3.72	2.77	Not determined	Not determined
Fe_2O_3	2.07	0.96	10.41	5.70	5.27	1.00	4.65	. . .	. . .
FeO	12.72	10.77	10.55	10.65	6.49	4.73	14.11	11.98	26.03
MnO	3.43	7.16	14.91	7.71	7.81	2.05	3.84	5.40	4.62
CaO	47.34	51.00	31.00	48.42	47.36	50.76	40.64	46.84	42.05
MgO	6.01	3.01	2.08	2.05	1.67	4.00	2.43	4.09	6.83
V_2O_5	1.19	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .
Ca	0.51	n. d.	n. d.	n. d.	n. d.	0.84	0.69	0.34	0.36
S	0.41	n. d.	n. d.	n. d.	n. d.	0.67	0.55	0.27	0.29

(3) *Gases.* Analyses, see *ante.*

(4) *Fumes.* In converting, some lime is blown out of the nose of the converter with sparks of iron, and with iron and manganese fumes.

(5) *Odds and Ends* are similar to those of the acid process.

SECTION 63. **Cost.** Nau[2] gives the following estimate of cost of producing pig iron and basic Bessemer steel in Alabama.

Items.	Quantities, in tons.	Price per ton of material used.	Price per ton of pig iron.
Ore	2½	$0.65	$1.63
Coke	1.15	1.80	2.07
Limestone	¾	0.35	0.26
Labor	. . .	. . .	1.30
General expenses, taxes, incidentals, etc. . . .	. . .	. . .	0.40
Repairs, tools, oils, etc. .	. . .	. . .	0.45
Relining furnace	. . .	. . .	0.20
Total	. . .	. . .	$6.31

Items.	Cost price.	Weight per ton of ingots.	Cost per ton of ingots.
Iron (1.18)	$6.31	2,640	$7.446
Ferromanganese	60.00	15	0.450
Coal	1.00	375	0.266
Dolomite	5.00	50	0.125
Wood	. . .	. . .	0.005
Lime (18 %)	3.00	400	0.600
Tar (about 10 % on burnt dolomite)	.12 gallon	¾ gallon	0.090
General expenses, repairs, taxes, etc.	. . .	. . .	0.750
Labor	. . .	. . .	1.500
Total	. . .	. . .	$11.232
Less value of basic slag	. . .	. . .	0.600
Total per ton of 2,240 pounds .	. . .	. . .	$10.632

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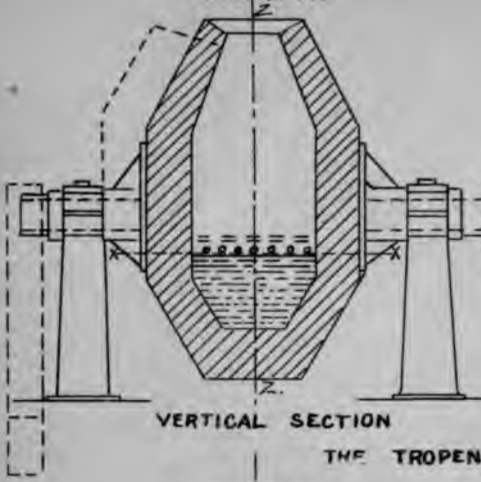
[1] Ledebur, "Manual," 4th ed., p. 1020.

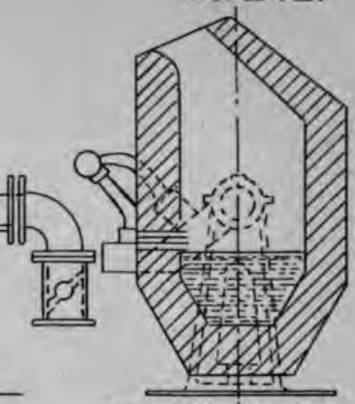
[2] "Iron Age," May 3, 1894.

FIG. 241.



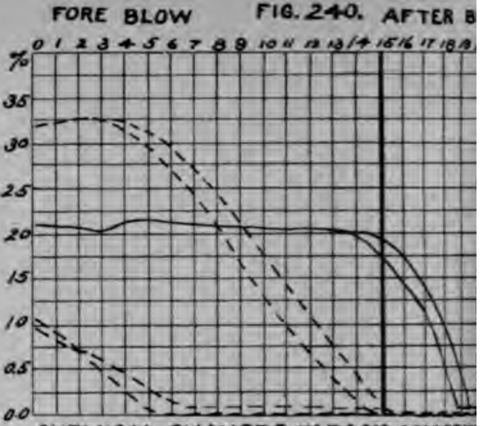
VERTICAL SECTION

FIG. 242.



VERTICAL SECTION ZZ.

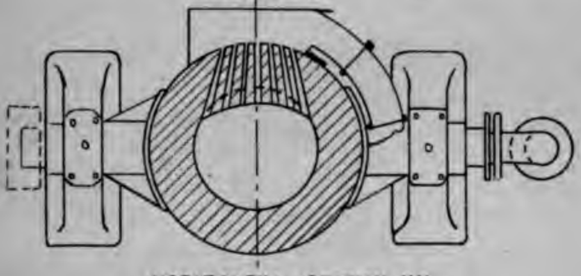
FIG. 240.



CHEMICAL CHANGES IN BASIC CONVERT

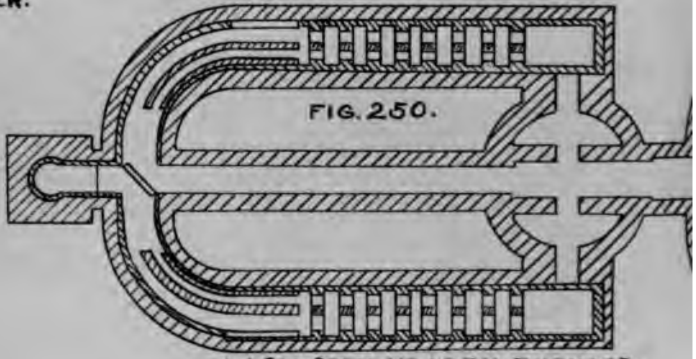
THE TROPENAS CONVERTER.

FIG. 243.



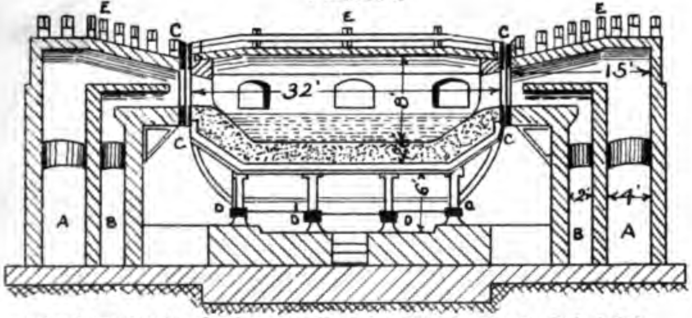
HORIZONTAL SECTION XX.

FIG. 250.



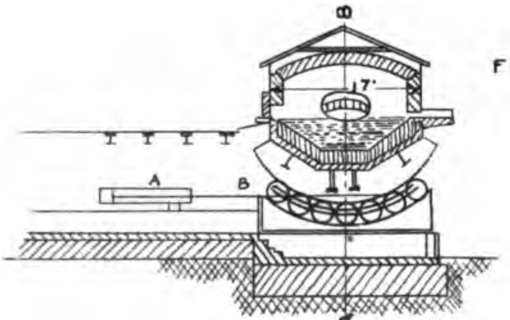
LASH OPEN HEARTH FURNACE.

FIG. 251.



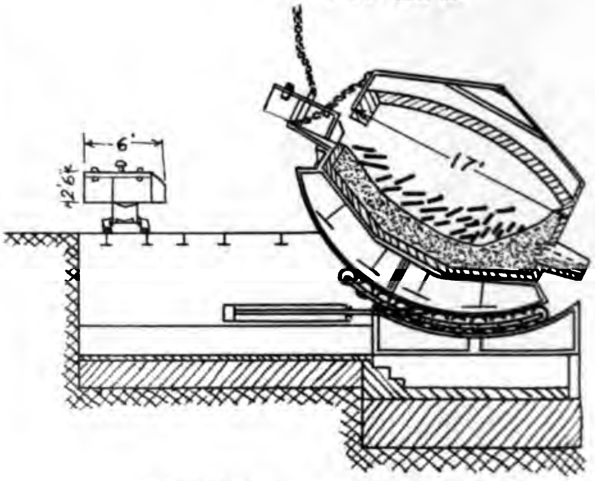
LONGITUDINAL SECTION THROUGH LINE A B FIG. 252.

FIG. 252



CROSS SECTION.

FIG. 253.



CHARGING OF FURNACE.

CAMPBELL REVOLVING OPEN-HEARTH FURNACE. FIGS. 251 - 253.

C. THE BABY BESSEMER CONVERTER.

Literature. General: Howe, "Journal Iron and Steel Institute," 1886, II, p. 651. Hardisty, "Revue Universelle des Mines," 1887, XXI, p. 414 ("Berg- und Hüttenmännische Zeitung," 1888, p. 26). Vogel, "Zeitschrift des Vereins deutscher Ingenieure," 1892, p. 406; "Berg- und Hüttenmännische Zeitung," 1892, p. 281; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1893, pp. 44, 57; "Stahl und Eisen," 1898, p. 183. Rott, *op. cit.*, 1901, p. 999; "Zeitschrift des Vereins deutscher Ingenieure," 1900, p. 144. Simonson, "Iron Trade Review," July 30, 1903. Lilienberg, "Foundry," 1904, XXIII, p. 260; "Iron and Steel Magazine," I, p. 276. Stoughton, "Transactions American Institute of Mining Engineers," XXXIV, contains many references and tabulated lists of the steels produced.

Clapp-Griffith Converter. Howe, "Steel," p. 356. Witherow, "Transactions American Institute of Mining Engineers," XIII, p. 745; XIV, p. 919. Hunt, *op. cit.*, XIII, p. 753; XIV, p. 139. Stoughton, *op. cit.*, XXXIV.

Robert or Walrand-Delattre Converter. Howe, "Steel," p. 357; "Engineering and Mining Journal," August 30, 1890. Bildt, "Berg- und Hüttenmännische Zeitung," 1891, p. 167. Garrison, "Journal Iron and Steel Institute," 1889, II, p. 266. Robinson, "Iron Age," October 31, 1889. Pourcel-Valton, "Revue Universelle des Mines," 1891, XIII, p. 146. Walrand, "Stahl und Eisen," 1887, p. 390.

Tropenas Converter. Powell-Colné, "Iron Age," May 7, 1896. Pixis, "Engineering and Mining Journal," May 1, 1897. Vogel, "Stahl und Eisen," 1898, p. 193. Adams, "Iron Trade Review," August 8, 1901. Tropenas, "Journal American Foundrymen's Association," 1901, X, p. 109. Simonson, "Iron Trade Review," October 29, 1903. Stoughton, "Transactions American Institute of Mining Engineers," XXXIV.

Long-tuyère (Stoughton) Converter. Stoughton, "Transactions American Institute of Mining Engineers," XXXIV.

SECTION 64. **In General, Clapp-Griffith, Robert, Tropenas, Long Tuyère (Stoughton), etc.** Baby Bessemer converters are small vessels treating from 0.8 to 3 tons of pig iron at a time; they usually have had an acid lining, and the blast is introduced from the side[1] instead of the bottom. The advantages of using such small side-blown converters are: cheapness of plant; smallness of blowing engine on account of low-pressure blast; casting of small ingots rolled in one operation to a market product, or making of small steel castings;[2] good quality of steel;[3] long life of bottom. The disadvantages are: the expense per ton of product and the conversion loss, which is 4–5 per cent. higher than in the bottom-blown large converter.

The leading converter at one time was that of

(1) *Clapp and Griffith*, a high, side-blown, stationary vessel, with a slag spout at a level high enough above the bottom that the slag may run off during the boil, and a tap hole for the steel in the side at the bottom. The first converter in this country was built in 1884; in 1888 fourteen were in use; in 1900[4] there were none.

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[1] Howe, "Steel," p. 342.

[2] Walrand-Legénisel's addition of ferrosilicon to increase temperature: "Stahl und Eisen," 1892, p. 117; 1893, p. 830; 1896, p. 704: "Journal Iron and Steel Institute," 1894, I, p. 26; 1896, II, p. 104; "Engineering and Mining Journal," April 14, September 15, 1894.

[3] Howe, "Steel," p. 71.

[4] "Engineering and Mining Journal," March 17, 1900.

(2) *The Robert, or Walrand-Delattre Converter* is rotary; the blast is introduced through horizontal tuyères placed fan-shaped in order to give the metal bath a rotary motion. At the beginning of a blow the air is forced in just under the surface of the metal bath, and as the blow progresses the converter is turned upward in order to lower the level of the tuyères. In 1899 steel was made in one Robert converter only, to be used for castings, while four

(3) *Tropenas Converters* (Figures 241–243)[1] were in operation in 1899 and thirteen in 1900. This is a tilting converter, with two rows of tuyères placed on one side. The lower circular "fining" tuyères blow *upon* the surface of the metal bath (not *into* it); the upper oblong "combustion" tuyères serve to burn the CO(H), generated by the lower tuyères, to $CO_2(H_2O)$, and thus to assist in keeping the steel hot and fluid. The pig iron[2] best suited for the converter has the following range of composition: Si, 2.50–3.50; Mn, 0.50–1.25; C, 3.00–4.50; S, 0.03–0.06; P, 0.04–0.07. The high percentage of Si is necessary to have the metal hot, much hotter than in the bottom-blown Bessemer converter. In working, four periods may be distinguished. During the first, the desiliconizing period, lasting about one-half of the total time required, the upper tuyères are kept closed, and the converter is tilted sufficiently forward that the jets of air passing through the lower tuyères at a pressure of 4–4.5 pounds shall strike the metal about halfway across. With the burning of the C begins the second period. The converter is somewhat righted, that the tuyères, having still a downward pitch, blow below the top level of the slag. The valve of the upper tuyères is now opened, at first partly, then wholly, to burn CO and H. After a partial drop of the flame begins the third period. This is distinguished by a fluctuating flame and dense volumes of red smoke. The last period begins with the large drop of flame, characteristic of the elimination of C, when the iron is deoxidized and recarburized by the addition of red-hot ferrosilicon and ferromanganese, either in the converter or the ladle. The time of the blow proper is from fifteen to twenty-two minutes. Between blows some time is spent in running out the steel, running in the melted pig iron, skimming off the slag, and placing tuyères. The combined cupola and converter loss is given at from 14 to 19 per cent., but often this percentage is raised considerably by the tuyères having become worn down too much and their position not changed to suit new conditions. Edgar Allen and Company, Sheffield, England,[3] produced with two converters, and five blows in one hour and fifty-five minutes, enough steel to make a 6-ton casting, the steel from a blow ranging from 3,000 to 3,200 pounds. In steel castings there are added toward the end of a blow 1 per cent. of ferrosilicon (14 per cent. Si) and 1.8 per cent. ferromanganese (75 per cent. Mn) to heat, quiet, and recarburize the steel. The pressure of the blast is 4½ pounds, the loss in conversion 16–19 per cent.

(4) *The Long-tuyère* (*Stoughton*) *Converter.* The main differences between this converter and the preceding one are: that it is longer, to correct excessive spitting, has only one row of long, pitching tuyères at the level of the metal (the upper row having been found unnecessary), and a movable bottom, which reduces the number of vessels required and cheapens repairs. A comparison of the subjoined analyses shows how effectively the CO is burned to CO_2 in the Stoughton converter: —

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[1] "Iron Age," May 7, 1896.

[2] Stoughton, "Transactions American Institute of Mining Engineers," XXIV.

[3] "Iron Age," May 7, 1896.

ANALYSES OF SIDE-BLOWN STOUGHTON CONVERTER GASES.

Time after beginning of blow.	Analyses—Per Cent.				Calculations from Analyses—Per Cent.			
	CO	CO_2	O	N and H	Total O	N[1]	O entering.	O burning Si, Fe and Mn.
4 minutes, flame starts . . .	0.0	8.2	1.1	90.7	7.1	88.7	23.6	16.5
10 minutes, boiling	0.3	24.3	0.4	75.0	18.3	73.0	19.4	1.1
12 minutes, shortening[2] . . .	0.4	8.8	0.2	90.6	6.8	88.6	23.5	16.7
17 minutes, after first drop . .	10.7	13.0	0.2	76.1	15.8	74.1	19.7	3.9
21 minutes, end of blow . . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .	. . .

[1] H being estimated as 2 per cent.

[2] *i. e.*, just before the first drop. There are two drops to the flame in this operation, the second marking the end.

ANALYSES OF BOTTOM-BLOWN CONVERTER GASES.

Time after starting blow.	Per Cent.					Reference.
	CO	CO_2	O	H	N	
2 minutes	. . .	10.71	0.92	. . .	88.37	Bell, "Principles," p. 390.
4 minutes	3.95	8.59	. . .	0.88	86.58	
6 minutes	4.52	8.20	. . .	2.00	85.28	
10 minutes	19.59	3.58	. . .	2.00	74.83	
12 minutes	29.30	2.30	. . .	2.16	66.24	
14 minutes	31.11	1.34	. . .	2.00	65.55	
18 minutes	End of blow.		. . .	. . .	. . .	
3 to 5 minutes	. . .	9.127	4.762	. . .	86.111	Tamm, "Journal Iron and Steel Institute," 1875, II, p. 669.
9 to 10 minutes	17.555	5.998	1.699	0.908	73.840	
21 to 23 minutes	19.322	4.856	0.967	1.120	73.735	
26 to 27 minutes	14.311	1.853	0.550	1.699	81.587	
2 to 3 minutes	. . .	6.608	7.256	. . .	86.137	
8 to 10 minutes	15.579	5.613	1.296	1.112	76.400	
12 to 15 minutes	25.580	4.144	0.980	1.040	68.256	
17 to 19 minutes	25.606	2.995	1.318	1.120	68.961	

Pig iron with 1.6 per cent. Si works satisfactorily; a converter treats 20–30 tons metal in five hours; a bottom lasts ten to twelve heats; the combined cupola and converter loss is 15–17 per cent.

SECTION 65. **Bessemer Steels.**

GRADES AND COMPOSITION.[1]

J. W. CABOT.

Grade.	C	P	Si	Mn	S maximum.	Cu	Elastic limit.	Tensile strength.	Elongation.	Reduction of area.
							Pounds per square inch.	Pounds per square inch.	Per cent.	Per cent.
Sheets (black, galvanized, or tinned)	0.10	0.10	0.01	0.35	0.08	. .	39,000	65,000	26	46
Wire rod, wire nails, cut nails	0.12	0.10	0.01	0.40	0.08	. .	39,300	65,500	26	45
Bridge rivets	0.15	. . .	. . .	. . .	. . .	. .	42,000	70,000	23	42
Bessemer axles	0.18	. . .	. . .	0.60	. . .	. .	46,000	75,000	21	36
Beams, angles, tees	0.15	. . .	. . .	. . .	. . .	. .	48,000	78,000	20	. . .
Bessemer bridge	0.20	0.08	. . .	0.75	. . .	. .	50,000	80,000	19	30
Machinery	0.25	. . .	. . .	. . .	. . .	. .	51,000	83,000	19	28
Toe-calk	0.28	. . .	. . .	0.80	. . .	. .	54,000	90,000	19	26
Light rails, 56 pounds . . .	0.32	0.10	0.05	0.90	0.08	. .	57,000	95,000	18	25
Medium rails, 85 pounds . .	0.45	0.08	0.08	0.85	. . .	. .	60,000	105,000	10	15
Heavy rails, 100 pounds . .	0.60	0.06	0.10	0.70	. . .	. .	. . .	. . .	8	. . .

Soft Bessemer steel used for sheets should not have more Mn than enough to make it roll smoothly. Wire-rod steel must be harder in order to stand the draught in the rod rolling mill. Steel for light rails has about 0.32 per cent. carbon; in heavy rails the C is increased to 0.60 per cent. In order that this may stand the drop test, the P must be reduced and also the Mn. If, however, Mn is reduced below 0.75 per cent., the ingot may not roll well. This is improved by increasing Si.

D. THE OPEN-HEARTH PROCESS.

Literature: Campbell, "Transactions American Institute of Mining Engineers," XIX, p. 218; XX, p. 229; XXII, p. 345; "Manufacture and Properties of Iron and Steel." New York, 1903.

SECTION 66. **In General.**[2] In the open-hearth process steel is produced by melting down a charge in a gas-heated reverberatory furnace having Siemens regenerative chambers. The process was first successfully carried out in 1865 by Martin Brothers (Sireuil, France), who

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[1] Richards, "Notes on Iron," p. 127.

[2] Monell, General History: "Iron Trade Review," November 14, 1901. Wellman, Early History in the United States: "Iron Age," December 5, 1901. Garrett, Open-hearth Plant in General: "Iron Trade Review," March 5, 1903. Christie, Modern Development: "Iron Age," August 2, 1902. Daelen, In General: "Stahl und Eisen," 1904, p. 507.

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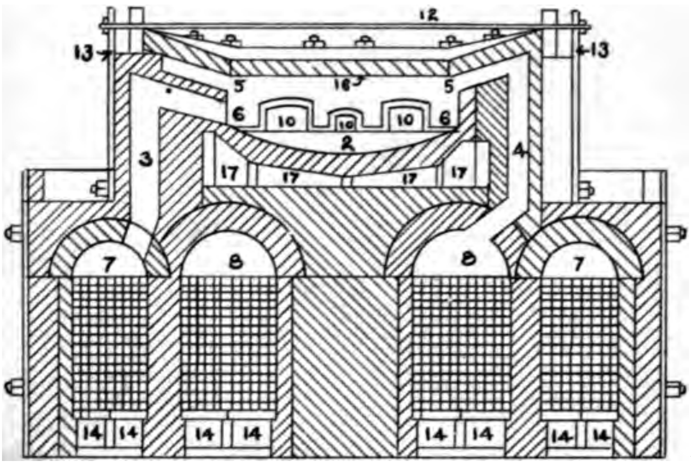
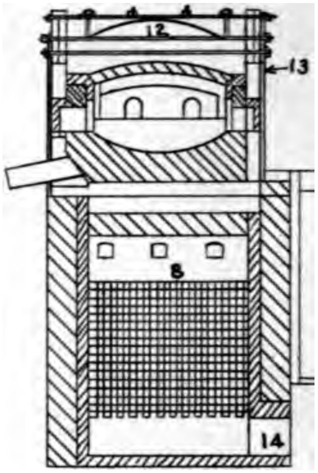
ASTOR, LENOX AND
TILDEN FOUNDATIONS.

FIG. 246.



LONGITUDINAL SECTION.

FIG 248.



CROSS SECTION.

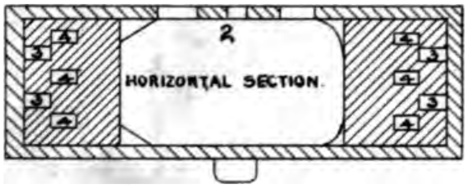


FIG. 205

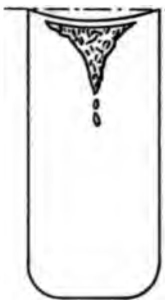
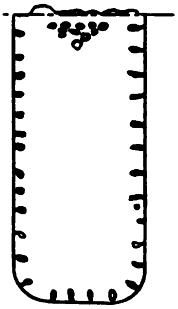


FIG. 206



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SECTIONS OF POROUS

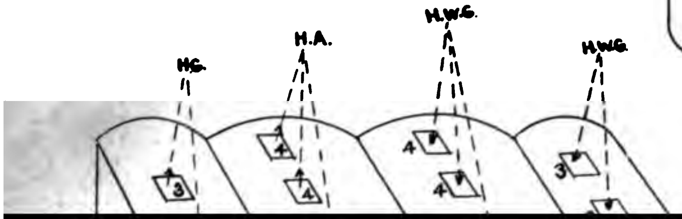

FIG.

melted down low-C wrought iron and high-C pig iron and obtained the medium-C steel. Previous trials in the same line had failed on account of the impossibility of maintaining a heat of sufficient intensity; the addition of the Siemens regenerative chambers to the reverberatory furnace furnished Martin Brothers the means. The process is called in France and Germany *Martin* or *Siemens-Martin Process;* in England and the United States, *Open-hearth Process.*

There are three modifications of the open-hearth process: —

(1) *Pig and Scrap Process.* Pig iron and wrought and steel scrap are charged into the furnace and melted down. The high C of the pig iron becomes diluted by the low C of the wrought iron forming steel. During the melting down considerable C, Si, and Mn becomes oxidized, so that the original wrought iron of Martin can be replaced by steel scrap (crop ends from Bessemer ingots, etc.).

(2) *Pig, Scrap, and Ore Process.* A lack of wrought iron or steel scrap to dilute the C of the pig iron can be made up by the use of rich iron ore, the O of which oxidizes C, and at the same time Si and Mn.

(3) *Pig and Ore Process.* This dispenses entirely with scrap. C, Si, and Mn are oxidized by the O of the ore and by a hot, steady oxidizing atmosphere.

The pig and scrap process is not much used today on account of the high price of scrap, while the pig, scrap and ore process is common in this country. Pig and scrap are first melted down and yield a metal bath low in Si and Mn but high in C; this is then oxidized by the addition of iron ore. The pig and ore process is rarely used on account of the slowness of the operation and small capacity, the consequent high consumption of fuel and cost of labor, and the imperfect reduction of the iron in the ore.[1]

There are two modifications of the open-hearth process, according to the composition of the pig iron and the character of the hearth material, whether acid or basic: The Acid and the Basic Open-hearth Processes.

SECTION 67. **The Open-hearth Furnace in General.** F. Toldt, "Regenerativ-Gasöfen," Leipsic, 1898. Lürmann, "Stahl und Eisen," 1890, pp. 10, 194. Saltz, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1893, pp. 1, 17, 30. Daelen, *op. cit.*, 1892, p. 12. Daelen-Pszczolka, *op. cit.*, 1901, p. 50.

A. *Stationary Furnaces.* Figures 244–246[2] represent two vertical and one horizontal sections of the common type of open-hearth furnace heated with producer gas; Figures 247–248[3] give perspective views of the regenerative chambers and of the roads, air, gas, and products of combustion travel, supposing the furnace to be in normal working order. In Figure 247, cold gas (*cg*) and cold air (*ca*) enter through flues, 20 and 19, respectively, above the butterfly valves (15); travel at the left through 14, and ascend in the checkerwork (9) of the gas (7) and air (8) regenerative chambers; take up the heat stored there;[4] leave the chambers superheated (*hg* and *ha*) through the flues 3 and 4, and, meeting at their terminations, 5 and 6, burn. The flame passes over the hearth (2) and leaves through flues 3 and 4 at the right. The hot waste

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[1] Ehrenwerth, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1886, p. 622; Schmidhammer, "Stahl und Eisen," 1902, p. 651.

[2] "Transactions American Institute of Mining Engineers," XXII, p. 354.

[3] Richards, "Notes on Iron," p. 104.

[4] Decomposition of CO in Checkers: "Stahl und Eisen," 1903, pp. 447 (Würtenberger), 693 (Ledebur).

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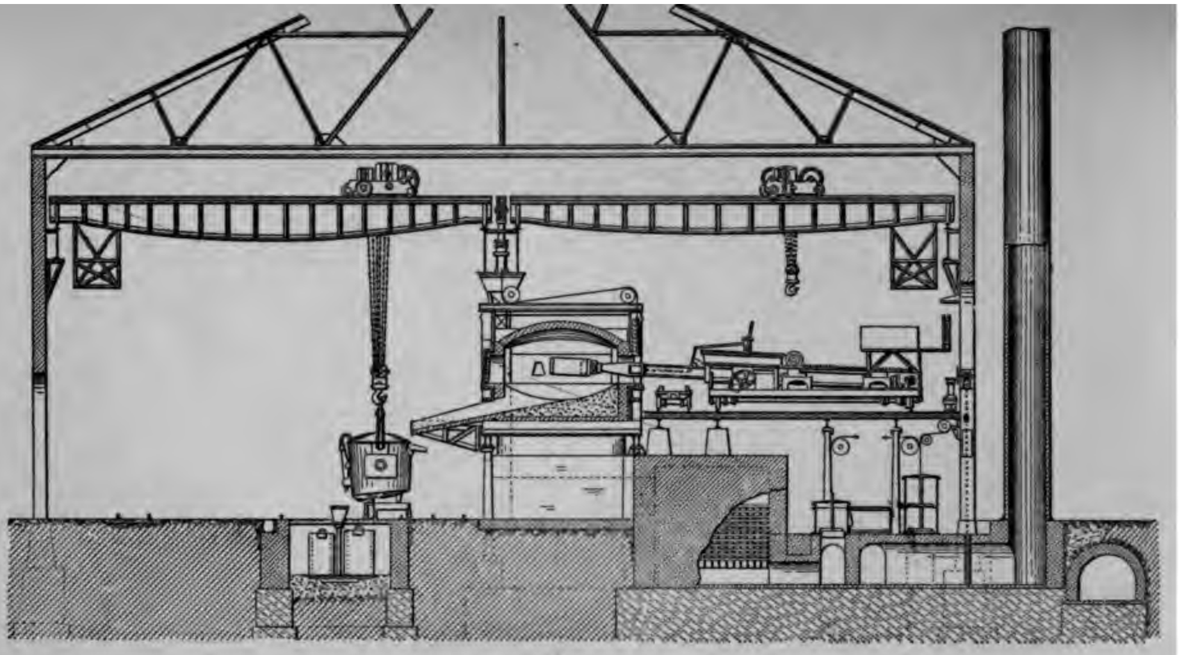

Fig. 249. Wellman Stationary Open Hearth Furnace.

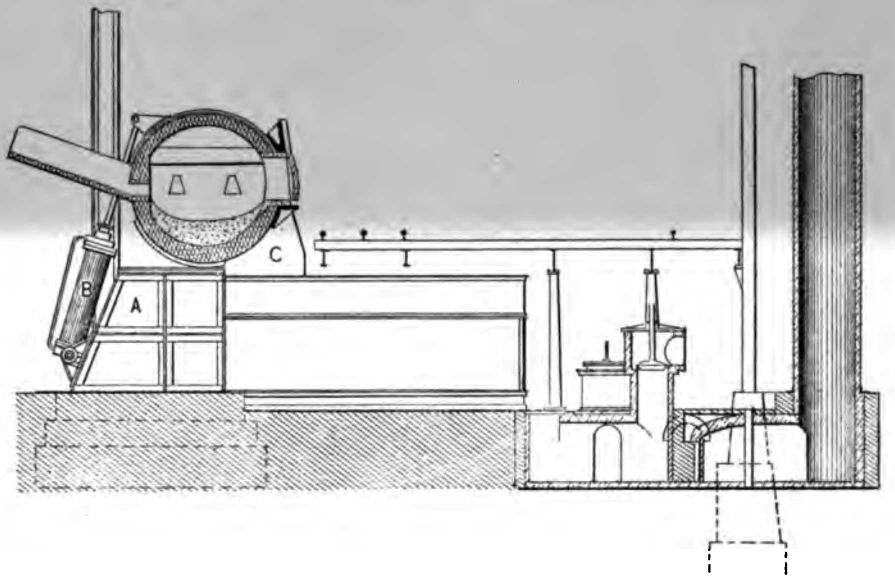

Fig. 254 Wellman Tilting Open Hearth Furnace.

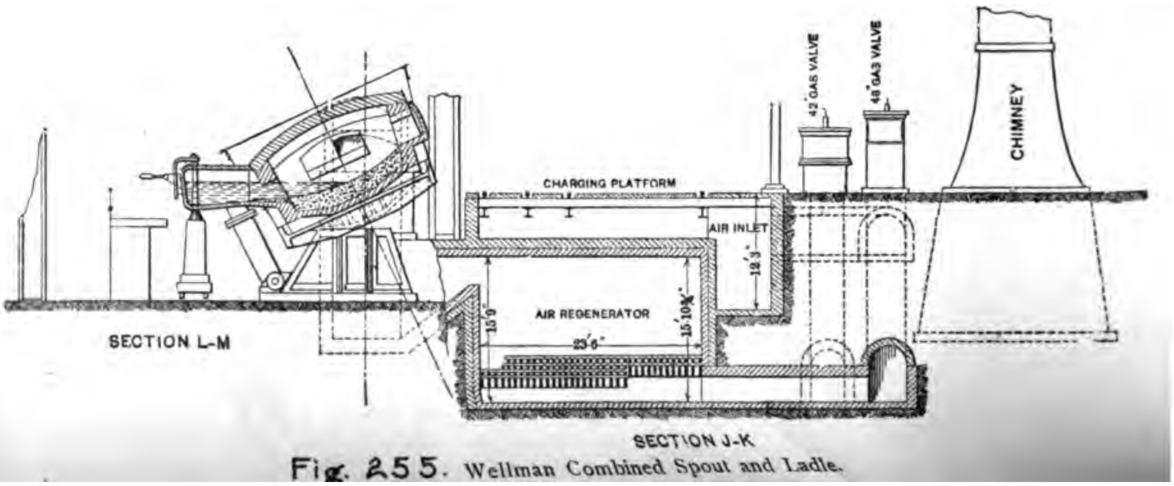

Fig. 255. Wellman Combined Spout and Ladle.

gases (*hwg*), descending through the checkerwork (9) of the regenerative chambers 7 and 8 to the right, give up their heat, leave them through 14, and, passing under the butterfly valves 15, enter the flue 18 leading to the stack. After one-half to one hour, valves 15 are reversed, and gas and air now travel in the opposite direction, from right to left. With every reversal of the gas and air current, the temperature will be raised, because the amount of heat stored in the checkerwork is being constantly increased, and because a nearer approach is made to perfect combustion with a smaller excess of air over the theoretical amount. The absorption of the heat by the melting of the charge and the loss by radiation usually prevent overheating; it is, however, necessary to regulate the admission of gas and air so that the regenerators shall not be heated above 1,000° C. Le Chatelier's measurements[1] gave the gas leaving the regenerator five minutes after reversing the valves 1,220° C. and the air 1,070° C.

In starting a cold furnace, it is necessary to heat the brick chambers, say with a wood fire, and fill the passages with waste gases before admitting producer gas and air, as the producer gas does not readily kindle and explosions are liable to occur.

The disadvantage of this common type of furnace is that the hearth rests on arches which become weak by expansion and contraction caused by the changes of temperature incidental to the constant reversal of the valves. *Wellman* (Figure 249)[2] took the vertical regenerating chambers from under the hearth and placed them beneath the charging floor, air and producer gas entering at the bottom and rising through the checkerwork on their way to the hearth. He thus furnished the means of giving the hearth firm and readily accessible foundations. The figure gives also the general arrangement of an open-hearth plant with 20-ton furnaces. The furnaces are built in a row. On the tapping side is the casting pit containing the molds (here bottom-casting) and the ladle suspended from an overhead (usually electric) traveling crane. In more modern plants,[3] the pit is done away with and the molds are placed on the main floor of the building (Figure 255). While this is more convenient, it is more dangerous in case of breakouts. On the charging side is the Wellman charging machine.[4] The charging boxes arrive on a train of buggies, are taken up one after another by the machine, raised, inserted through the doors into the furnace, and emptied by turning over. In the original article are given two horizontal sections and one longitudinal section of the furnace. Several years before, *Batho*[5] had removed the checkerwork from underneath the furnace and placed it at the ends.

In order to reduce the height required by vertical regenerators, necessitating the digging of pits to receive the chambers or raising all the stock to the elevated working floor, *Lash* constructed horizontal chambers. Figure 250[6] represents a horizontal section through the air flue and the two regenerative chambers for air and products of combustion. There are no gas chambers, as natural gas is used, which cannot be superheated without being decomposed. The disadvantages of horizontal chambers are: (1) that the hot gases will pass along the upper row

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[1] Howe, "Engineering and Mining Journal," October 11, 1890, p. 428.

[2] "Iron Age," July 25, 1895.

[3] "Cassier's Magazine," 1900, XVII, p. 270.

[4] Head, "Journal Iron and Steel Institute," 1897, I, p. 89; "Iron Age," August 27, 1896; Wellman, "Transactions American Institute of Mining Engineers," XIX, p. 313.

[5] "Journal Iron and Steel Institute," 1887, II, p. 119; "Transactions American Institute of Mining Engineers," XVI, p. 705.

[6] "Transactions American Institute of Mining Engineers," XVI, p. 697.

of passages and not give up their heat to the full section of the checkerwork; and (2) that there is a sluggish current, as, excepting in the vertical uptake leading into the hearth, there is no propelling action upon the gases in the horizontal chambers, when compared with the rising of the gas in the vertical chambers. These difficulties are overcome, however, by the use of a blower for the air current and by working the producer with undergrate blast. With producer gas a double set of regenerators is necessary.

B. *Tilting Furnaces.*[1] In order to facilitate the removing of the steel from the furnace just at the moment when it has the right composition, and to lighten the work, open-hearth furnaces have been made tilting, which allows the metal to be poured instead of being tapped. Further advantages are that slag can be poured off at intervals during the melting process (important in the pig and ore process, where much slag is made), and that metal desiliconized and partly decarburized on an acid hearth can be readily transferred onto a basic hearth to be dephosphorized and partly desulphurized, or the start is made in a basic furnace and the steel finished in an acid furnace.[2] The disadvantages of tilting furnaces are the greater initial cost, the larger cost of repairs, and the cost of the power. It may be noted that the Carnegie companies have adhered to the stationary furnace.

The Campbell Revolving Furnace (Figures 251–253)[3] was erected at the works of the Pennsylvania Steel Company, Steelton, Pennsylvania, in 1889. The hearth is placed upon rockers and revolves around its own axis on four roller bearings running in circular paths. It is operated by a hydraulic cylinder. The ends of the furnace and the fixed gas and air parts have cast-iron water-cooled plates *C*. The brickwork forming the ports is enclosed in an iron framework, so that when the arches are worn out the ports as a whole can be exchanged (C. E. Stafford). The furnace has horizontal regenerators. The producers are worked with undergrate blast, and air is blown into the chambers with an electric fan. When the furnace is in its normal position, the discharge hole, filled with loose sand, is above the level of the slag. In the ordinary Siemens furnace, the charge is introduced by placing it upon a paddle and sliding into the furnace. Formerly the Campbell furnace was tilted to an angle of 30° from the horizontal, and the boxes containing the charges were raised by a hydraulic crane from the buggies (Figure 253) on which they arrived, and the contents shot in through the charging doors. The hearth was made especially strong on the back, which, being baked in, formed a continuation of the hearth. Now the Wellman charging machine (Figure 249) has been introduced. It has become the standard of the country.[4] The furnace shown in the figures is a 45-ton furnace. A charge consists of 80 per cent. pig iron, the rest is scrap, with some ore. At the Pennsylvania Steel Company's Works, cold pig iron and scrap as well as direct metal are worked. A furnace makes fourteen heats a week.

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[1] Campbell, "Manufacture and Properties of Iron and Steel," New York, 1903, p. 206; Howe, "Mineral Industry," 1898, VII, p. 417; Head, "Journal Iron and Steel Institute," 1899, I, p. 69; Eyermann, "Stahl und Eisen," 1900, p. 310; "Journal Iron and Steel Institute," 1902, I, p. 259.

[2] Pennsylvania Steel Company, Steelton, Pennsylvania.

[3] "Transactions American Institute of Mining Engineers," XXII, p. 360; "Journal Iron and Steel Institute," 1899, I, p. 69; Campbell, "Manufacture and Properties of Iron and Steel," p. 210.

[4] Charging Machines: "Journal Iron and Steel Institute," 1897, I, p. 89; "Stahl und Eisen," 1891, p. 305; 1895, pp. 669, 950; 1897, pp. 708, 857, 1042; 1900, pp. 748, 996; 1902, p. 806; 1903, p. 829; 1904, p. 642; Dingler, "Polytechnisches Journal," CCCVII, p. 108.

The Wellman Tilting Furnace.[1] Figure 254[2] shows a vertical section through the centre of the furnace. It is barrel-shaped, and rests with its two segmental steel castings on strong steel standards with horizontal upper surfaces. The furnace is rolled or tilted forward or backward by two nearly vertical hydraulic cylinders mounted on trunnions. The rolling surfaces are provided with rack and pinion, which keep the furnace in position without, however, having to support any weight. On account of the furnace's tipping or rolling when it is being charged or when the back wall (the one on the discharge side) is being baked, the connection with the ports is broken. This is a decided disadvantage as against the Campbell furnace. The Wellman furnace has the advantage of having no machine-finished work.

At Ensley, Alabama, ten 50-ton Wellman furnaces were put up in 1898 in which the steel is poured through a combined pouring spout and ladle, as shown in Figure 255.[3] This does away with the overhead traveling crane.

The New Siemens Furnace has been described by Head and Pouff.[4] The principle of the regeneration of part of the waste gas, on which the furnace is based, is discussed by Åkerman.[5]

The cost of an open-hearth plant in dollars, excluding rolling department, may be approximated by multiplying the capacity of tons pig iron with 1,500–2,000. Treat[6] gives a detailed estimate of cost of a basic open-hearth plant with two 50-ton furnaces. The total is $230,000.

References for illustrated descriptions of the leading American plants are subjoined: Duplex Plant;[7] Illinois Steel Company, South Chicago, Illinois;[8] Carbon Steel Company, Pittsburgh, Pennsylvania;[9] Pennsylvania Steel Company, Steelton, Pennsylvania;[10] Granite City Steel Company, Granite City, Illinois;[11] Worth Brothers, Coatesville, Pennsylvania;[12] Apollo Iron and Steel Company, Vandergrift, Pennsylvania;[13] Buhl Steel Company, Sharon, Pennsylvania;[14] Homestead, No. 2 Plant, Carnegie Steel Company, Pittsburgh, Pennsylvania;[15] Homestead, No. 3 Plant, Carnegie Steel Company, Pittsburgh, Pennsylvania;[16] Open-hearth Casting Plant, Alliance, Ohio;[17] Duquesne Plant;[18] American Tube and Stamping Company,

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[1] Head, "Journal Iron and Steel Institute," 1899, I, p. 70; "Iron Age," July 25, 1895; "Cassier's Magazine," 1900, XVII, pp. 268, 270; "Mineral Industry," VII, p. 417.

[2] "Iron Age," *l. c.*

[3] "Journal Iron and Steel Institute," 1899, I, p. 80, Plate X; "Iron Age," May 25, 1899.

[4] "Iron and Steel Institute," 1889, II, p. 256; "Iron Age," 1892, XLIX, p. 1037; "Engineering and Mining Journal," 1889, XLVIII, p. 408; "Stahl und Eisen," 1900, p. 616; "Bulletin de l'Industrie Minérale," 1891, V, p. 273; "Berg- und Hüttenmännische Zeitung," 1892, LI, p. 70.

[5] "Journal Iron and Steel Institute," 1890, I, p. 24.

[6] "Iron Age," March 19, 1903.

[7] *Op. cit.*, June 4, 1891.

[8] *Op. cit.*, July 25, 1895.

[9] *Op. cit.*, March 9, 1893.

[10] *Op. cit.*, March 28, 1895.

[11] *Op. cit.*, April 16, 1896.

[12] *Op. cit.*, June 11, 1896.

[13] *Op. cit.*, June 17, 1897.

[14] *Op. cit.*, December 30, 1897.

[15] *Op. cit.*, August 2, 1897.

[16] *Op. cit.*, June 30, 1898.

[17] *Op. cit.*, September 25, 1902; "Iron Trade Review," September 25, 1902; "Foundry," 1901, XXI, p. 182.

[18] "Iron Age," January 1, 1903.

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[illegible]

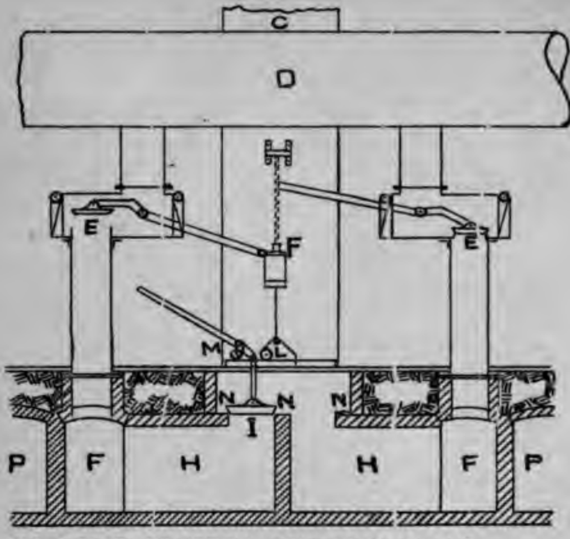


FIG. 256-7 CAMPBELL REVERSING VALVE.

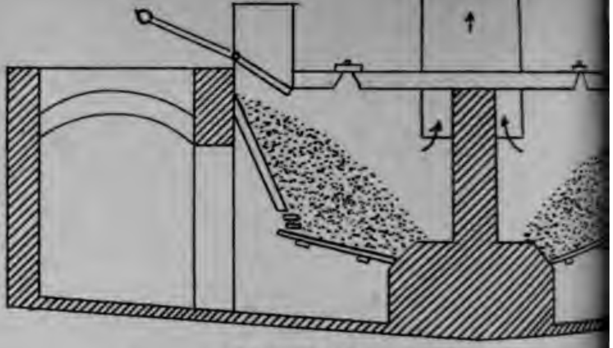
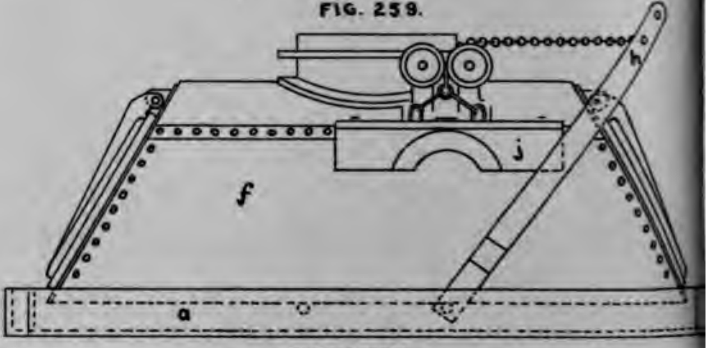


FIG. 260.
SIEMENS GAS PRODUCER.

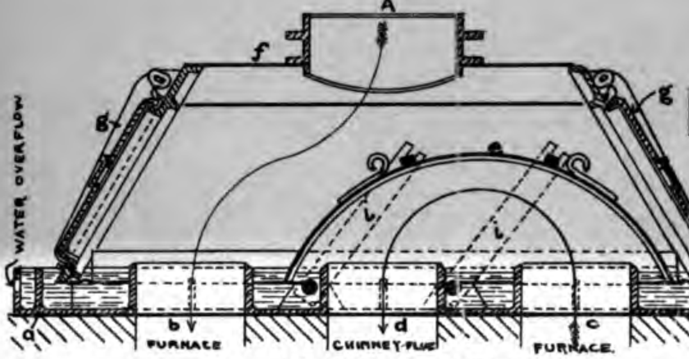
FIG. 259.



ELEVATION.

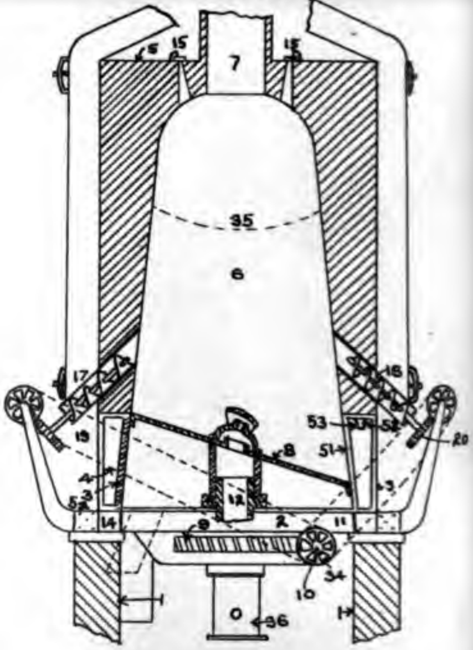
THE FORTER WATER SEAL REVERSING VALVE.

FIG. 258.



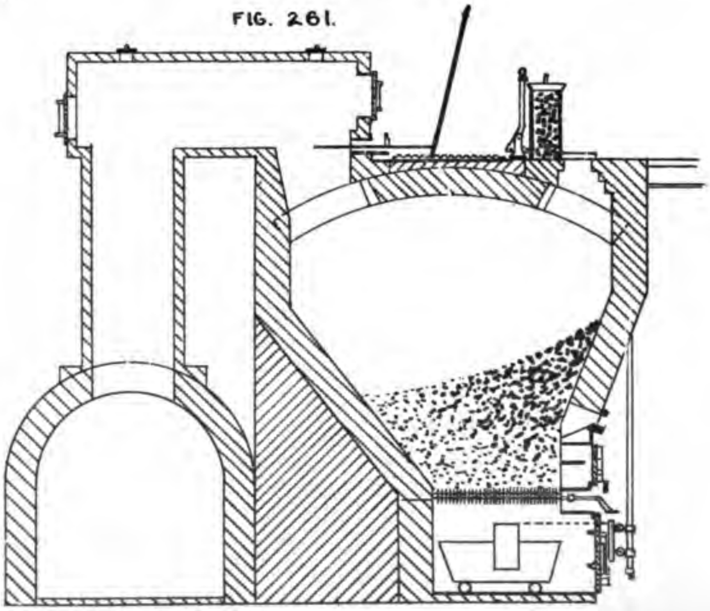
LONGITUDINAL SECTION.

FIG. 266.



KITSON GAS PRODUCER

FIG. 261.



SWINDELL-SIEMENS GAS PRODUCER WITH WATER SEAL.

Bridgeport, Connecticut;[1] Ensley, Alabama.[2] Further: Illies, American Open-hearth Plants;[3] Langheinrich, American Open-hearth Plants and Rolling Mills.[4]

SECTION 68. **The Open-hearth Furnace in Detail.**[5] (1) *Checkerwork.* Grüner[6] has shown that the draught of a chimney is at its maximum when the gases have a temperature of 273°, or in round figures at 300° C.; also that there is only a very little difference in the draught, whether the gases enter the base of the chimney at 200 or 300° C.; hence the checkerwork ought to be large enough to cool the gases leaving the hearth at 1,600–1,700° C. down to 200° C. Le Chatelier's measurements[7] gave 1,500° C. as the temperature of the furnace at the moment of casting. Of course, the larger the regenerator the more slowly will it give up the heat stored in it, and the less frequent need be the reversing of the valves. The main point in the checkerwork is to expose the necessary surface of refractory material (specific heat, 0.25–0.23) to the gas current. Grüner[8] calculates that with producer gas a pair of regenerators should contain 50 kilograms brickwork for each kilogram of coal gasified between two reversals of the valves. Generally 60 kilograms are given with one hour between two reversals of valves. Now 60 kilograms fire brick (specific gravity, 1.8) have a surface of 0.033 cubic meter, and, as only half of the surfaces form flues, 2 x 0.033 = 0.066 cubic meter per kilogram coal, or 2,115 cubic feet of surface per short ton coal gasified per hour will be required, or approximately 1 cubic foot per pound coal per hour. The cross section[9] of the regenerators must be large to prevent the gases from rushing through them. For every net ton of bituminous coal gasified per hour a pair of regenerators has a cross section 250–300 square feet. If 1 kilogram bituminous coal is burned with 20-25 per cent. excess of air over the theoretical amount, it yields about 11 cubic meters products of combustion at 0° C., or 48.3 cubic meters at 1,200° C. With 100 kilograms (220 pounds) coal per hour and a temperature of 1,200° C. of the products of combustion, 1.3 cubic meters (46 cubic feet) of gases will pass through 1.24 square meters (13 square feet) of open spaces in the checkerwork at a rate of 3.28 feet per second. The cross section of the gas chamber is made smaller than that of the air chamber, viz., 3 : 4 or 2 : 3, although theoretically the volume of air required is approximately the same as that of the producer gas. The reason for this is that the gas entering the regenerator is warm, while the air is cold.

(2) *The Valves.*[10] The Siemens Butterfly Valve was represented in connection with the Ingot Soaking Furnace (Figures 213–216) and the Richards perspective drawing of the Siemens regenerators (Figures 247–248). This valve is still frequently used on account of the simplicity and the ease with which it is manipulated, although it has a decided defect in that it is alternately

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[1] "Iron Age," November 5, 1903.

[2] "Stahl und Eisen," 1900, p. 364.

[3] *Op. cit.*, 1902, pp. 645, 713.

[4] *Op. cit.*, 1901, pp. 1169, 1220, 1294; 1902, p. 106.

[5] Hess, "Proceedings Engineers' Club of Philadelphia," XXI, pp. 48–65; Abstract, "Engineering Review," 1904, X, p. 359.

[6] "Traité de Métallurgie," Paris, 1875, I, p. 283.

[7] Howe, "Engineering and Mining Journal," October 11, 1890, p. 428.

[8] "Traité de Metallurgie," I, p. 383.

[9] Ledebur, "Manual," 4th ed., p. 158.

[10] Williams, "Iron Age," April 7, 1904.

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exposed on one side to the cool incoming gases and on the other to hot products of combustion, and becomes warped and leaks. Water cooling to some extent remedies the defect, but not wholly.

The Campbell Valve[1] is shown in Figures 256–257. There is only one reversing valve, *I*, which, being suspended from the buggy *M*, is carried from one water-cooled seat, *N*, to the other and remains always under the same conditions, viz., in contact with air above and with producer gas below.

The Forter Valve,[2] shown in Figures 258–259, resembles in principle the slide valve of a steam engine. It consists of a base or trough casting, *a*, filled 2½ inches with flowing water and having three openings, *b*, *c*, and *d*, leading to and from the regenerative chambers and to the chimney; of a heavy steel plate hood *e* serving as valve, and a housing *f* with end doors *g* and inlet pipe *A*. In the drawing the hood covers the furnace flue *c* and the chimney flue *d*; the waste gases, having given up their heat to the regenerators, pass into the chimney flue, while producer gas (respectively air) entering at *A* passes into the other furnace flue *b* leading to the opposite regenerators to be heated. The valve is reversed by means of the lever *h*; the hood is raised out of the water seal by the arms *ii* and dropped again into the seal on the opposite side, thereby connecting the furnace flue *b* at the left with the chimney flue *d* and uncovering the furnace flue *c* at the right and connecting it with the gas (respectively air) inlet *A*. The counterweight *j* balances the weight of the hood. The valve has found much favor; it suffers, however, as do all water-seal valves, from the fact that the gas coming in contact with water carries much steam into the furnace. Forter constructed an improvement[3] of this valve in which he changes the gas currents by raising and lowering the seal water.

Other forms of valves are described in "Stahl und Eisen," 1885, p. 344; 1886, p. 668 (Ehrenwerth); 1895, pp. 268 (Schmidhammer), 388 (Huth); 1903, pp. 333, 404 (Poetter), 456 (Turk), 691 (Nägel, Chantraine), 738 (Czekalla), 891 (Fischer), 1275 (Agrimont). "Iron Age," May 16, 1889 (Weiss); May 7, 1903 (Poetter); July 23, 1903 (Czekalla); September 3, 1903 (German); September 10, 1903 (Schild); April 24, 1904 (Wight and Hyatt). "Journal Iron and Steel Institute," 1892, I, p. 238 (Wailes). "Iron and Steel Magazine," 1904, I, p. 169 (Agrimont). "Iron Trade Review," May 7, 1903 (Poetter); May 26, 1904 (Fleischer). Toldt, "Regenerativ Gasöfen," p. 236.

One hundred pounds coal burned with 20 per cent. excess air required 10,200 cubic feet air at 0° C. and 760 millimeters pressure. If burned in one hour, the air at 30° C. having to pass the flue per second will be 3.14 cubic feet. The area of the air valve for 100 pounds coal per hour will be 0.31 square foot. One hundred pounds coal yielding 7,000 cubic feet producer gas will require the gas valve to have an area of 0.26 square foot (Hesse, *l. c.*).

(3) *The Ports.*[4] These are usually so arranged that the gas enters below the air. The reasons for this are that the gas being lighter rises and becomes thoroughly mixed with the air, that the metal will not be exposed to the strongly oxidizing effect of the hot air, and that the roof will not be burned out so quickly. The ports usually pitch toward the hearth that the flame

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[1] Campbell, "Manufacture and Properties of Iron and Steel," 1903, pp. 214 and 215.

[2] "Iron Age," March 17, 1898.

[3] *Op. cit.*, December 18, 1902; "Stahl und Eisen," 1903, p. 166.

[4] Schmidhammer, "Stahl und Eisen," 1894, p. 751; Toldt, *op. cit.*, p. 288.

may strike the charge; gas and air ought to meet about 5 feet from the metal (Campbell), so that the gas may begin to burn before it strikes the charge; if the distance is smaller, some of the gas will burn in the checkerwork and melt it. Both gas and air should enter the hearth under pressure; the work of the draught of the furnace is to prevent the flame from striking the roof and to draw the products of combustion through the checkerwork. Gas and air enter the hearth either through single or divided ports. The area of the air port is 7 square feet, of the gas port 6 square feet for every net ton of bituminous coal burned per hour.[1] Hesse (*l.c.*) gives 0.59 square foot for air and 0.4 square foot for gas for each 100 pounds coal burned per hour, which per ton of coal is equal to 11.8 cubic feet for air and 8 cubic feet for gas. Toldt[2] and Ledebur[3] say that the velocity of air and gas should not exceed 26 feet per second; an average is 16½ feet. The ends of the ports are of silica brick.

(4) *The Stack.* The area varies, of course, with the size of the furnace. One hundred pounds bituminous coal converted into producer gas and burned in the furnace yield 36,500 cubic feet stack gases at 260° C. and 760 millimeters pressure. With a velocity of 16½ feet per second, 0.6 square foot area are necessary for each 100 pounds coal per hour. Hesse (*l. c.*) found the stack area from ten open-hearth furnaces to average 0.95 square foot. The smallest height is probably 50 feet; it reaches 100–150 with large furnaces.

(5) *The Hearth and Roof.* The hearth is oblong; it is built upon iron plates supported by I-beams so as to permit air-cooling. Water-cooling, which would lengthen the life of the bottom, is too powerful, causing sculls to form. With small furnaces, length : width = 2 : 1; with large furnaces = 3 : 2; the width should not exceed 15 feet on account of the difficulty of reaching across, as there are doors only on the front (charging) side. The depth of the metal[4] in a 5-ton furnace is 12–15 inches; in 30- to 50-ton furnace, 18–24 inches; the shallower the bath the stronger is the oxidation and the quicker the rate of melting. The larger the amount of slag formed (acid pig and ore process and basic process) the shallower can be the hearth, as the metal is protected from excessive oxidation by the heavy layer of slag. Measured at the sill of the furnace, the hearth area of furnaces with over 15 tons capacity is 6¾ square feet per ton of product with acid furnaces and 7¾ square feet with basic furnaces.[5] Measured at the surface of the metal bath, the area is from 5 to 6 square feet per ton of metal charged, assuming the usual depth of 15–18 inches (Hesse, *l. c.*).

The working bottom is monolithic. In an acid furnace[6] there is placed on the iron shell a 9-inch course of fire brick; the furnace is then fired up to an approximate working temperature, when a layer of sand (97.5 SiO_2, 1.8 Al_2O_3) or quartz crushed to pea size[7] and mixed with 2–5 per cent. clay is spread to the thickness of about 1 inch and heated for about ten minutes until it has fritted and come to a set; it is followed by a second layer, and so on until it has reached a thickness of 18–24 inches. In the basic furnace, calcined dolomite, ground

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[1] Ledebur, "Manual," 4th ed., p. 163.

[2] "Regenerativ Gasöfen," p. 296.

[3] *Op. cit.*, p. 164.

[4] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 122.

[5] Ledebur, "Manual," 4th ed., p. 1026.

[6] "Transactions American Institute of Mining Engineers," XXII, p. 397.

[7] Ganister from Connellsville, Pennsylvania: SiO_2, 98.26; Al_2O_3, 1.17; Fe, CaO, MgO, H_2O, 0.57. "Iron Trade Review," May 2, 1901.

and mixed with tar, is rammed with hot irons directly onto the iron shell or onto a layer of magnesia brick, the iron shell or the top of the course of brick being first painted with tar. It is tamped down in 2-inch layers and brought to a set, for which purpose 5 per cent. basic slag had been mixed in. Sometimes the bottom is put in as a whole. In this case the bottom is covered with a layer of burnt dolomite so as to prevent the tar from being burned out when the furnace is being fired up; the tar ought to be distilled, leaving behind a coke skeleton acting as a bond.[1] A bottom, if well made and patched regularly, is good for 1,000 and more heats. The sides of the disk-shaped working bottom reach well up towards the sides of the furnace carrying the roof. The roof, the sides and ends of all open-hearth furnaces are made of silica brick (98 per cent. SiO_2, 1.5 per cent. CaO, 0.5 per cent. others). The roof is usually horizontal. Formerly it was given a pitch downward to force the flame to strike the charge; experience taught that this construction shortened the life of the roof. The other extreme is to arch the roof, and this is found still with many furnaces. The roof is always made thin (9 inches) in order that the heat lost by radiation may keep it comparatively cool. In a basic furnace, the joint where acid roof and basic hearth meet is made of chromite brick. It is advisable to protect the joint by ramming hearth material in front of it. The sides are about 13 inches thick. On the front (the charging side) are two to four doors, according to the size of the furnace. On the back is the tap hole of the stationary furnace, which is at the lowest point of the bottom. It should be large, say 6–8 inches square, so as not to offer any obstructions in tapping. It is rammed full of refractory mixture, which must fulfill the requirements of not breaking out, of allowing a bar to be driven through it, and of not being eroded during the flow of the metal. In acid furnaces, sometimes fusible sand is first rammed in and is followed by infusible sand; in basic furnaces, the inside of the tap hole is rammed with tar and burnt dolomite or tar and burnt magnesite, the outside with sand. With rotating furnaces, the tap hole remains open or is loosely filled with sand, as it is above the level of the slag, except when pouring.

SECTION 69. **The Fuel of the Open Hearth.** The leading gas used in the open-hearth furnace is producer gas. In some favored localities *natural gas* is used, with 87–97 per cent. marsh gas (CH_4), the rest being mainly N and small amounts of H, CO, CO_2, C_2H_4, and O. The gas has a high calorific power, and is introduced into the furnace without passing through the regenerators, as in superheating it is in part decomposed, depositing soot. If burned with air superheated in the usual way, it gives all the heat that is necessary. Open-hearth furnaces using natural gas, therefore, have only two regenerative chambers (for air) instead of the four required with producer gas (two for air, two for gas).

Producer Gas[2] is the result of imperfect combustion of C by means of air. It is made in a producer, generally a low, shaft-like furnace, holding a deep bed of incandescent fuel, through which air or air and steam is drawn or forced. According to Ernst,[3] C heated in air begins to be oxidized at 400° C. and burns mainly to CO_2; the formation of CO_2 increases with the temperature up to 700° C., when that of CO grows up to 995° C.; above this temperature

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[1] "Stahl und Eisen," 1887, p. 853; 1890, p. 222.

[2] Campbell, "Transactions American Institute of Mining Engineers," XIX, p. 136; Lürmann, "Stahl und Eisen," 1903, pp. 433, 515, 1151, 1154.

[3] "Journal Iron and Steel Institute," 1893, I, p. 217; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1894, p. 239; Hempel, "Berg- und Hüttenmännische Zeitung," 1896, p. 166.

no more CO_2 is formed. A producer run on carbon at 1,000° C. ought to furnish a mixture of CO and N. The composition of such a gas would be

Per cent.	CO	N	Calorific power.
By volume	34.57	65.41	1 cubic ft. = 30 cal.
By weight	34.59	65.43	1 pound = 382 cal.

In actual work, gas free from CO_2 is not obtained, as producers are not run throughout at 1,000° C.[1] and the fuel bed is not sufficiently high to prevent any free O from passing through it. This burns some CO. The gas contains further some H. This comes from the fuel and from the decomposition of the moisture of the air by incandescent carbon: H_2O and $C = H_2 + CO$.

The following averages of three analyses by Ebelmen, using charcoal[2] and coke,[3] represent a very high grade of gas from carbonized fuel: —

From	Per cent. by	CO	N	CO_2	H
Charcoal	volume	33.3	63.4	0.5	2.8
	weight	34.1	64.9	0.8	0.2
Coke	volume	33.6	64.2	0.7	1.5
	weight	33.8	64.8	1.3	0.1

It is not often, however, that carbonized fuels are used for making producer gas; usually natural fuels — wood, peat, brown coal, bituminous coal, and anthracite — serve for this purpose. In gasifying these, two processes go on simultaneously: distillation and imperfect combustion (see Figure 262); and the gas will contain more or less volatile C_xH_y. In the following analyses by Gantt,[4] the anthracite used contained 5 per cent. volatile C_xH_y, and 10 per cent. ash; the bituminous coal, 55 per cent. F. C. and 32 V. H-C. It is presupposed that none of the V. H-C has been allowed to be condensed. If the gas cools, as is always the case when producer and furnace are separate, some of the C_xH_y of the bituminous coal gas will be condensed and the gas will lose considerable part of its heat value. In fact, the two gases when cooled down will show approximately the same composition.

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[1] Le Chatelier's measurements gave 720° C. as the temperature of the gas leaving the producer, and 400° C. when in the cooling tube: "Engineering and Mining Journal," October 11, 1890, p. 429.

[2] "Annales des Mines," 1843, III, p. 215.

[3] *Op. cit.*, 1844, V, p. 81.

[4] "Cassier's Magazine," IX, p. 49.

No steam used.	CO	V. H-C	H	CO_2	N	Combustible.	Original energy in gas.	CALORIFIC POWER.	
								Per lb. gas.	Per cu. ft. gas.
Anthracite . . .	27.79	1.37	. . .	3.72	67.14	29.16	64.77	391	25.8
Bituminous coal .	23.67	11.88	. . .	4.01	60.44	35.55	77.53	616	44.7

Making producer gas is an exothermic process: 1 kilogram $C + O = CO + 2,473$ cal. In its formation 30 per cent. of the heat developed in perfect combustion (1 kilogram $C + O_2 = CO_2 + 8,080$ cal.) is obtained. If the gas is cooled down it loses these 30 per cent. and can develop only the remaining 70 per cent., or 5,607 cal., when burned to CO_2. The gas will be cooled down to some extent if producer and furnace are separate. There will be no loss of heat, excepting losses by radiation, etc., from the producer if this is placed close to the furnace and the gas can enter with its initial heat. As it is in many cases more economical to have the producers separate from the furnaces in which the gases are to be burned, part of the sensible heat of C burning to CO has been utilized by introducing water vapor or CO_2 into the producer. Water vapor acting upon incandescent C is decomposed into H_2 and CO or CO_2 according to the temperature of the C. The reaction taking place[1]

at and above 1,200° C. is exclusively: I. $H_2O + C = H_2 + CO$;
between 500° and 1,200° C. is mainly: II. $2H_2O + C = H_4 + CO_2$.

In producer gas, there will therefore be found both CO and CO_2. The theoretical gas, according to equation I, has the composition:—

	CO	H
Volume, per cent.	50.0	50.0
Weight, per cent.	93.3	61.7

According to $\underset{(18)}{H_2O} + \underset{(12)}{C} = 2H + CO$, 1 kilogram H_2O gasifies $\frac{2}{3}$ kilogram C; 1 kilogram $H_2O = \frac{1}{9} H + \frac{8}{9} O$. To decompose H_2O, $\frac{1}{9} \times 28,780$ cal. $= 3,198$ cal. are required; the $\frac{2}{3}$ C burning to CO give $\frac{2}{3} \times 2,473 = 1,649$ cal. The difference between the two values, $3,198 - 1,649 = 1,549$ cal., represents the endothermic character of the reaction. By introducing water vapor with air, the sensible heat of the CO is reduced, but the heating value of the gas is increased.

The theoretical gas, according to equation II, has the composition:—

	CO_2	H
Volume, per cent.	33.3	66.7
Weight, per cent.	91.7	8.3

According to $\underset{(36)}{2H_2O} + \underset{(12)}{C} = H_4 + CO_2$, 1 kilogram H_2O gasifies $\frac{1}{3}$ kilogram C. To decompose 1 kilogram H_2O, $\frac{1}{9} \times 28,780$ cal. $= 3,194$ cal. are required. The $\frac{1}{3}$ C burning to CO_2 gives $\frac{1}{3} \times 8,080 = 2,693$ cal. The difference between the two values, $3,194 - 2,693 = 501$ cal., shows that the reaction of equation II is less endothermic than that of equation I, while the heat value of the resulting gas is about the same.

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[1] Ledebur, "Gasfeuerungen," p. 69.

CO_2, acting upon incandescent C, is reduced to CO, viz., $\underset{(44)}{CO_2} + \underset{(12)}{C} = 2CO$. One kilogram CO_2 gasifies $\frac{3}{11}$ kilogram C. To reduce 1 kilogram CO_2 to CO, 1,529 cal. are necessary; the $\frac{3}{11}$ kilogram C, burning to CO, give $\frac{3}{11} \times 2{,}473 = 674$ cal.; hence the difference $1{,}529 - 674 = 855$ cal. shows the strongly endothermic character of the reaction. As pure CO_2 is not used on a large scale, but part of the products of combustion of a furnace which contain at least 79 volumes of N, the gas obtained in the producer will be correspondingly diluted. It can be shown that the resulting gas has about the same composition as that obtained by gasifying C with air, so that only the heat brought to the producer by the products of combustion would represent a saving over the use of cold atmospheric air. This can be better utilized by superheating the air. CO_2 is therefore little used in making producer gas,[1] while there is hardly a producer run at present without the admixture of steam, at least when the resulting gas is conducted first through pipes to the distant furnace where it is to be burned.

Steam used.	CO	V. H-C	H	CO_2	N	Combustible.	Original energy in gas.	CALORIES.	
								Per lb. gas.	Per cu. ft. gas.
Anthracite [1] . . .	30.24	1.49	9.68	4.02	54.57	41.41	83.14	521	36
Bituminous coal [1] .	24.00	12.60	8.60	5.30	49.50	45.20	87.80	800	53

[1] "Cassier's Magazine," IX, p. 49.

The volumes of CO_2 in both gases are high; they should not exceed 4 per cent.; the lowest is 2 per cent., the highest 8 per cent.

Producer gas can be made from all sorts of fuels, but some are better suited than others. Wood, peat, and brown coal are sometimes used, but, being very rich in H_2O, require special apparatus for condensing the water vapor. They retain their form while they are being gasified, shrinking slowly until they finally crumble to ashes.

Bituminous coal is the leading fuel. Splint coal, which falls to pieces, and coking coal, which fuses, swells, and cakes, are not desirable, as they both obstruct the passages of the gas. Gas, steam, and furnace coals, which cake only slightly and are rich in volatile matter, form the most desirable fuel. Nut coal is the most suitable, but run of mine, slack, or a mixture of the last two is generally used. The coal ought to run low in ash and water. Water vapor, if present in abundance in the gases, will, beginning at 600° C.,[2] have a decomposing effect upon CO, viz., $H_2O + CO = 2H + CO_2$. One ton Pittsburgh coal gives about 150,000 cubic feet of producer gas.

Anthracite, semi-anthracite, and brown coal have a tendency to decrepitate in the producer and thus choke the draught. Anthracite best suited is No. 1 buckwheat (over $\frac{3}{16}$–$\frac{3}{8}$ and under $\frac{3}{8}$–$\frac{5}{8}$ inch holes); it ought to run low in ash, which must be difficultly fusible, and high in volatile matter. Anthracite in the form of culm or poorly prepared buckwheat is not suited.

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[1] See new Siemens furnace, p. 173.

[2] Blass, "Stahl und Eisen," 1886, p. 4; Würtenberger, *op. cit.*, 1903, p. 447; Ledebur, "Gasfeuerungen," p. 14; Naumann-Pixtor, "Berichte der deutschen Chemischen Gesellschaft," XVIII, pp. 1647, 2724, 2894.

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ASTOR, LENOX AND
TILDEN FOUNDATIONS.

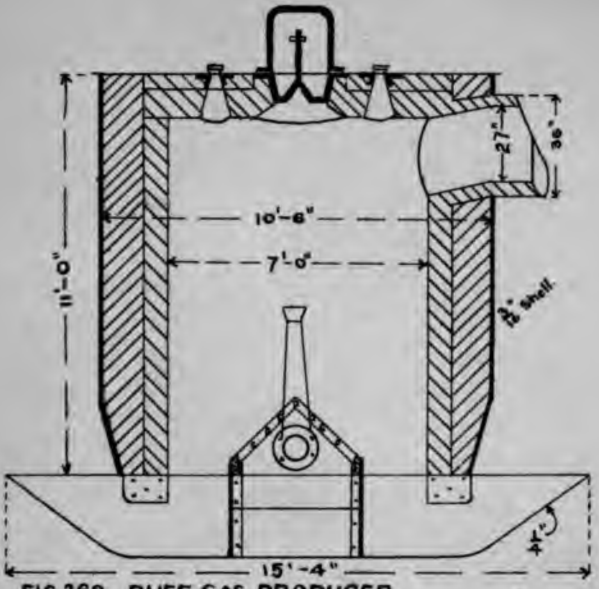
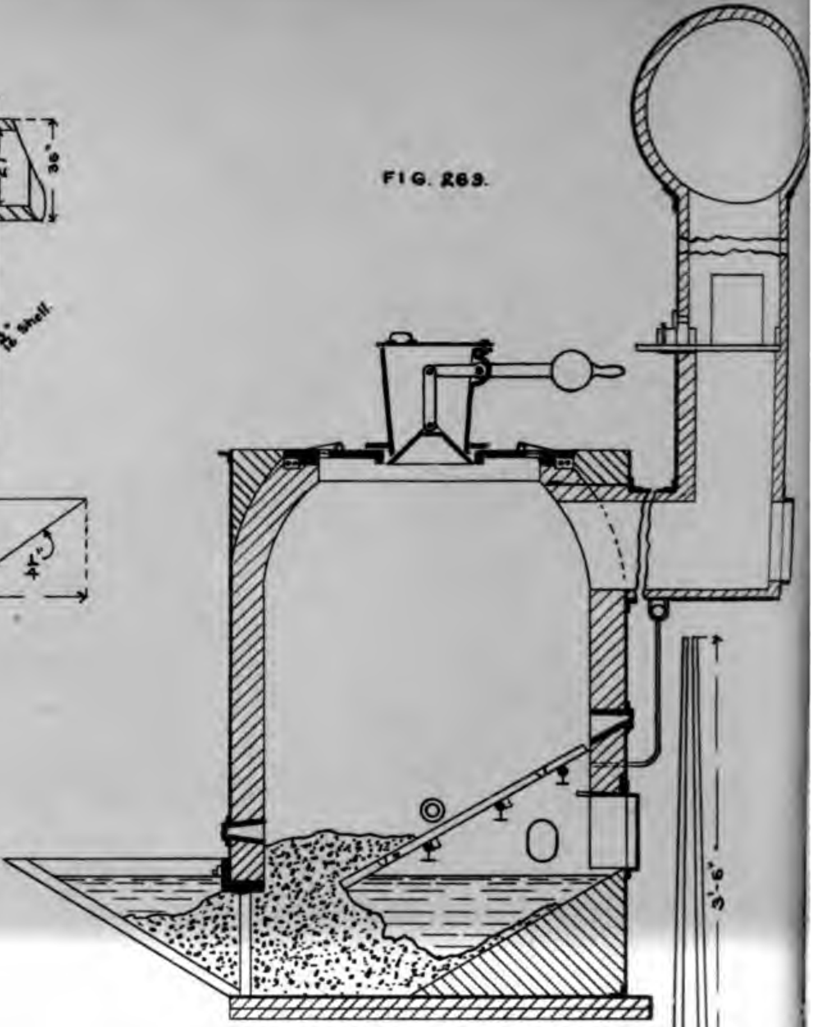


FIG. 268. DUFF GAS PRODUCER.

FIG. 269.



SMYTH GAS PRODUCER.

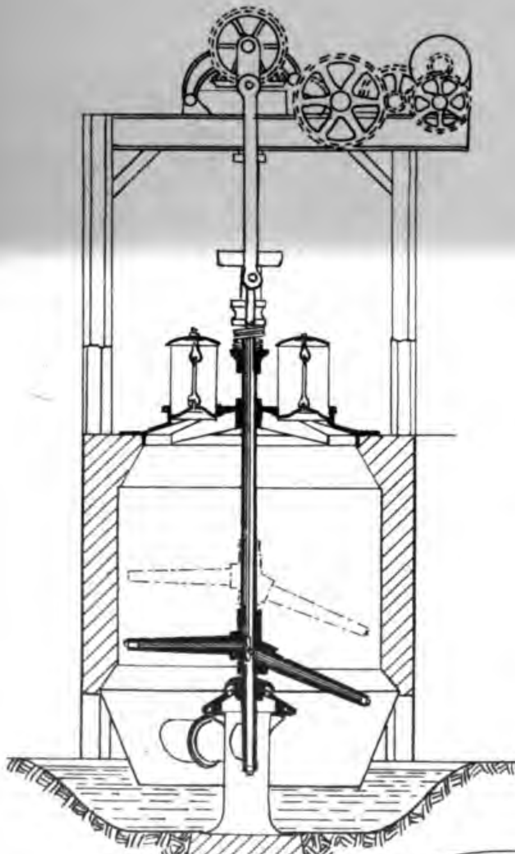


FIG. 270.

THE FRASER-TALBOT GAS PRODUCER.

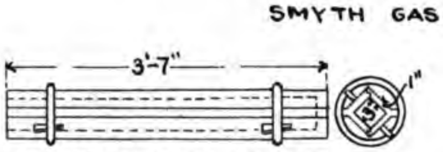
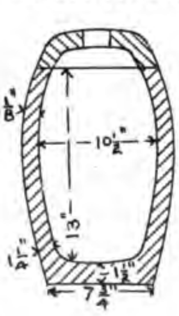


FIG. 279, SPLIT MOULD.

FIG. 278.



GRAPHITE STEEL CRUCIBLE.

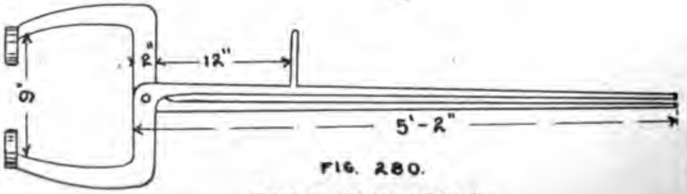


FIG. 280. PULL OUT TONGS.

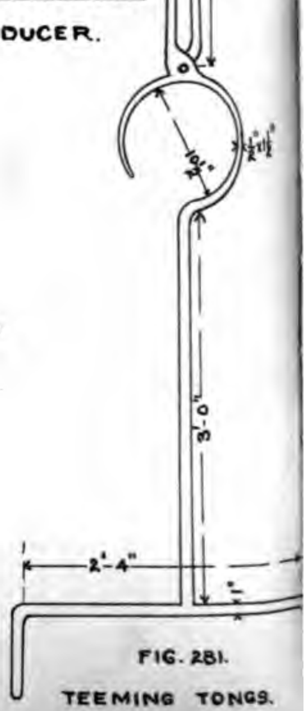


FIG. 281. TEEMING TONGS.

Exception: Kitson works culm at Scranton, Pennsylvania. One gross ton buckwheat anthracite gives about 170,000 cubic feet gas.

Charcoal and coke are not used in producers, being too expensive. As far as their behavior in gasifying is concerned they are very desirable, and, being practically free from volatile matter, the gas can be better piped than when made from natural fuels.

Baker[1] used the excess gas and the tar from by-product coking ovens to heat open-hearth furnaces at Sydney, Cape Breton.

SECTION 79. **Gas Producers.**[2] It is difficult, if not impossible, to find a classification that will include all the forms of producers that have been constructed. In all producers, with one exception, distillation and gasification take place in a single chamber; in the Gröbe-Lürmann[3] producer the two processes take place in separate chambers. Of the producers of the first class some are worked by natural draught, others by blast,[4] with or without steam; some have a horizontal or inclined grate, others have a solid bottom, which in some cases is stationary, in others movable; many modern producers have mechanical stirrers; some producers have a water seal, others are worked dry, etc. The producer may be placed close to or form part of the reverberatory furnace which it supplies with gas, or a number of producers may be grouped together and the gas conducted through one or more mains to the different furnaces of a plant.

In all single-chamber producers three zones (Figure 262) can be distinguished: the ash zone at the bottom, followed by the combustion, and the distillation zone. The function of the ash zone[5] is to heat air and steam. As the ashes are generally (excepting in the Kitson producer) not cleaned out continuously, they accumulate. This has to be kept in mind in regulating the depth of the fuel bed. The combustion zone occupies the greatest depth of the charge; the distillation zone comprises only the upper part of the fire.

The height of the bed of fuel in a producer varies with the fuel. It is the greater the lighter the fuel; thus wood will have 10 feet, peat 8 + feet, lignite and bituminous coal $3\frac{1}{2}$–$4\frac{1}{2}$ feet (sometimes, however, 6 feet), anthracite $2\frac{1}{2}$–3 feet. If with bituminous coal the fuel bed exceeds a depth of 6 feet, the lower part[6] is not thoroughly broken up, no matter how thoroughly the upper part may be stirred; the zone of combustion becomes honeycombed with large cavities, affording passage for undecomposed steam and air. This condition is most marked along the walls, and the intense heat produced on the interior surfaces of these coke-chimneys causes clinkers to fuse to the brickwork. Practice therefore demands that the thickness of the fire be limited, that steam be used to avoid extremes of temperatures, and that the mass be kept thoroughly broken by frequent poking.

Producers in this country are not often run exclusively by natural draught; as a rule a steam injector furnishes the blast, unless, of course, the ash is to be fused and tapped as a slag,

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[1] "Iron and Steel Magazine," 1904, I, p. 21.

[2] J. Deschamps, "Les Gazogènes," Paris, Dunod, 1902; Rowan, "Proceedings (English) Institute of Civil Engineers," 1886, LXXXVI, p. 2–113.

[3] Ledebur, "Gasfeuerungen," Leipsic, 1891, p. 60; Fischer, "Chemische Technologie der Brennstoffe," Brunswick, 1901, II, p. 295.

[4] Improved Blowers for Gas Producers: "Iron Age," March 6, 1902.

[5] "Transactions American Institute of Mining Engineers," XXII, p. 372.

[6] *Op cit.*, XXII, p. 373.

[illegible]

[illegible]

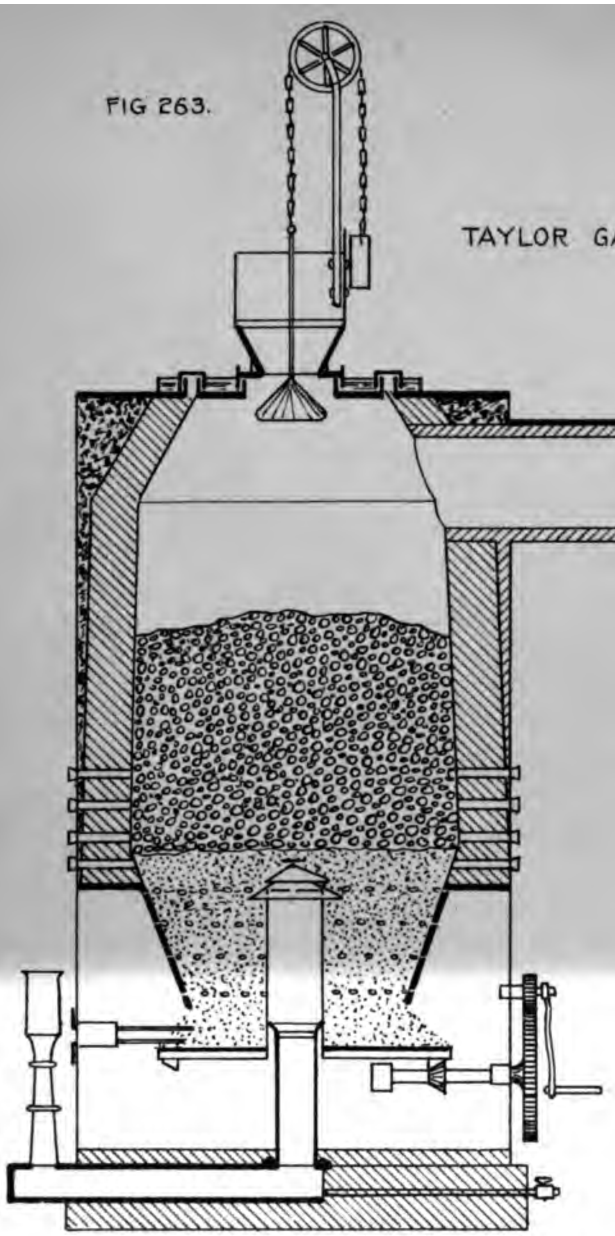
[illegible]

[illegible]

[illegible]

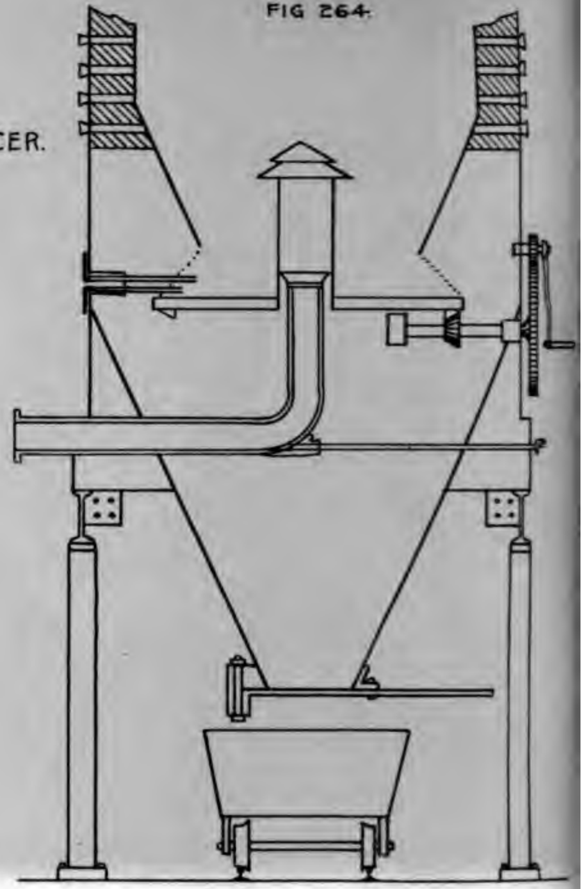
FIG 263.

TAYLOR GAS PRODUCER.



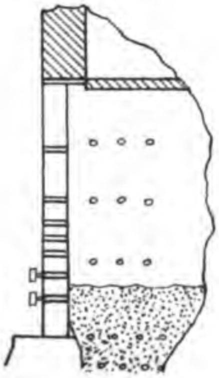
TYPE A.

FIG 264.



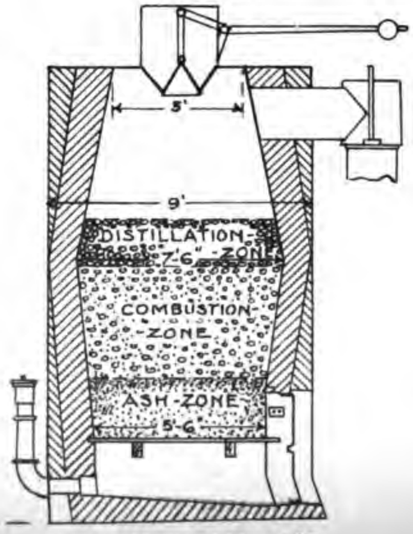
SECTION.

FIG. 265.



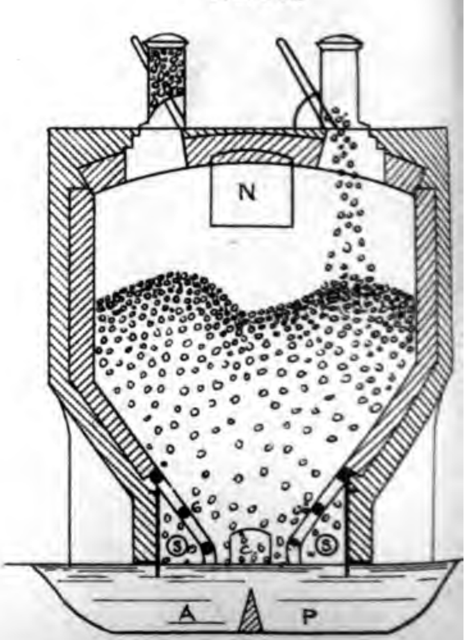
TYPE B.

FIG. 262.



THE WELLMAN GAS

FIG. 267.



SWINDEL GAS PRODUCER

when fans are employed. Producers are preferably run slowly, as the gas is richer in CO than when run quickly and hot. The gas leaves the producer at a temperature ranging 300–900° C.; 700° C. is perhaps an average figure.

(1) *The Siemens Producer* (Figure 260). This was the first commercially successful gas producer. It is a rectangular fire-brick chamber with one side inclined at an angle of 45–60° and provided at its foot with a step grate ending in the horizontal grate *B*. *A* are the charging holes, *E* the poking and peep holes; the gas passes off at *C*, its flow being regulated by the damper; four producers form a block; the gases from the four vertical flues unite in the main flue *D*. The area of each producer is at the grate 6 x 3 feet, and at the top 9 x 7 feet; the producer holds 10–12 tons coal and gasifies 2 tons per day. It is worked with natural draught without the use of steam; the gas is essentially a hot gas, which cools considerably in the gas main before it reaches the furnace where it is to be burned. The efficiency is therefore low. The original Siemens furnace has become obsolete. Where used today it is worked with a closed ash pit and a steam injector.

Below follow descriptions of some of the leading producers used today in this country, all of which work with a cinder grate and a deep bed of ashes.

(2) *The Swindell Gas Producer with Water Seal* (Figure 261)[1] is a modified Siemens producer with closed ash pit, steam injector, and shaking grate.

(3) *The Philadelphia Engineering Works, or Wellman Producer* (Figure 262).[2] This is a circular, wrought-iron shell, 14 feet high and 9 feet in diameter, lined with fire brick and red brick. The producer has a bosh, 7 feet 6 inches in diameter narrowing to 5 feet at the top. The bosh prevents air from creeping up at the walls. The charge rests on wrought-iron grate bars, the grate being 6 feet 9 inches x 6 feet 1 inch. The front is closed by two pairs of doors, one above the other. When drawing ashes, a false grate is put in above the bearing bars by pushing bars through the fire above the ashes; they will support the charge while the lower bars are being drawn and the ashes and clinkers dropped into the closed ash pit. While cleaning, the puppet valve closing the gas-outlet pipe is closed to prevent the gas from backing down through the fire onto the workmen. The coal is charged at the centre through bell and hopper; a steam injector below the grate furnishes blast and steam; the gas is drawn off at the side. Capacity = 4–5 tons per day.

(4) *The Taylor Gas Producer.* The characteristics of this producer are: (1) a circular, solid bottom which, on being revolved, breaks up any clinkers and discharges them with the ashes into the closed ash pit; (2) a deep bed of ashes; (3) a central steam-injector blast carried through the ashes to the fuel. The two leading types are shown in Figures 263–265. Both have a bosh. Type *A*, an iron shell lined with fire brick, is suited for anthracite and bituminous coal of which the ashes do not clinker readily. In type *B*, the lower part is a water jacket, the upper part an iron shell lined with fire brick. This producer is suited to coals the ashes of which clinker readily, as the clinkers will not adhere as easily to the jacket as to the brick wall. With both types, the part beneath the bottom plate is often hopper-shaped (Figure 265) to receive the ashes to be dropped at intervals into an ash car. The shaft is slightly contracted toward the top. The bottom of the bosh has a smaller diameter than the revolving bottom

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[1] "Cassier's Magazine," IX, p. 51.

[2] *Op. cit.*, IX, p. 52.

and does not reach down to it; the space between the two is filled with ashes, which, with an angle of repose of about 55°, glide over the edge of the bottom, when this is revolved. In order to grind up the ashes, four sets of agitating bars or scrapers are arranged above the table. In case the ashes should be discharged faster on one side than the other, with the result that the coal would sink more quickly on the discharging side, the bars can be withdrawn, which will retard the discharge. The ash bed, which is put on the bottom before firing up, is about 3½ feet deep and is kept about 6 inches above the cap of the central pipe. The bottom is revolved as soon as this height is exceeded, *i. e.*, every six to twenty-four hours according to the rate of working. Ashes are taken out once in twenty-four hours. The blast is furnished by a steam jet. In the side walls are four rows of peep holes to watch the process and the position of the top of the ash bed. The producer is made in seven sizes, Numbers 2 to 8; Number 2 is 2 feet in diameter, 10 feet high, and has a grate area of 3.1 square feet; Number 8 is 8 feet in diameter, 15 feet high, with a grate area of 50.3 square feet; Number 8 will gasify in twenty-four hours 6½ tons anthracite coal (pea) or 8 tons West Virginia or Pennsylvania bituminous coal. With bituminous coal, the fuel bed has to be deeper than with anthracite.

(5) *The Kitson Gas Producer* (Figure 266).[1] It consists of a circular, brick-lined iron shell, with inclined, continuously revolving iron bottom (8) to agitate the coal and discharge the ashes into a water box, and a continuous feed, which delivers the coal below the normal fuel line (35). The inside dimensions of a 7-foot producer are: diameter at bottom, 7 feet; at top, 4 feet; height, 12 feet. The iron shell rests on eight cast-iron (3) columns, 3 feet high, which are bolted to a circular base plate (2) supported by six 10-inch I-beams (14), resting with their ends on two 18-inch foundation walls (1). The inclined (18°) hearth (8) carrying the fuel bed is cast in one piece with the central hollow shaft (12), and is carried by a train of steel balls rolling in a groove upon the base plate. The shaft, carrying a horizontal worm wheel (9), is driven by an endless screw (11). The hearth, which is elliptical and 2 feet higher at one end than the other, has a grating or perforated plate (8) on the brick surface to facilitate the discharge of the ashes. The space in which the eight supporting columns stand is enclosed by upright cast-iron ash plates (3), which are radially adjustable to permit regulating the discharge of the ashes onto the base plate, whence they are conveyed by scrapers attached to the underside of the hearth to an opening in the base plate, and dropped into a water-sealed receiver extending sufficiently outside the producer to permit removing the ashes. The air and steam blast (36) enter the producer through the hollow central shaft, and are distributed through a central tuyère and the perforated hearth plates or gratings. The coal, which was formerly charged from the top by a bell and hopper feed (60 pounds every five to six minutes), is now introduced continuously below the fuel line by a screw feed (17 and 18), connecting by belting (19 and 20) with the shaft (11) of the endless screw driving the worm wheel of the shaft. The bottom makes about one revolution in forty minutes with anthracite, one in twenty to thirty minutes with bituminous coal. About 2 horse power are required to drive the bottom. The advantages claimed for the producer are that, by the slow, continuous revolving of the bottom, the fuel is always in motion, with the result that a uniform good quality of gas is obtained, and that no clinkers are made which require labor for barring.

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[1] "Iron Age," March 14, 1895; April 30, 1896, ill.; September 30, 1897, results; improved form: "Engineering and Mining Journal," February 26, 1898.

(6) *The Swindell Water-seal Steam-blast Producer* (Figure 267).[1] This is a circular producer with a large bosh; the continuation of the bosh walls is formed by two sloping grates, *G*; *C* is the cleaning door at the base of the grate. The steam-injector pipes, *S*, extending the entire length of the grate, have rows of small openings; the water-seal pan *P*, with central partition, has the width of the grate and a length equal to that of the diameter of the furnace; the gas-outlet pipe *N* is located at the highest point and has a cleaning door; it can be closed off from the gas main by the sand-damper plate. There are two coal hoppers; one is shown filled with coal, the other emptying the coal. In the roof are four ball poke holes and in the plates of the hopper two stopper poke holes, so that every part of the coal surface can be reached.

(7) *The Duff Water-seal Producer* (Figure 268).[2] This is a cylindrical gas producer with water seal. It has a roof-shaped central grate and an undergrate steam blast. The outer shell of $\frac{3}{16}$-inch steel has a diameter of 10 feet 6 inches and a height of 11 feet; it is lined with red brick and fire brick so as to give a square shaft 7 x 7 feet. The water-seal pan, 15 feet 4 inches long x 7 feet 0 inch wide x 2 feet 3 inches deep, is of $\frac{1}{4}$-inch steel. The gas-outlet pipe (36 inches outer diameter) is lined with brick (27 inches inner diameter). The top has a bell and hopper feed and the usual poking holes. Capacity claimed: 12 tons Pittsburgh coal in twenty-four hours, 1 ton coal = 150,000 cubic feet gas.

(8) *The Smythe Continuous and Self-cleaning Gas Producer* (Figure 269).[3] This is a circular producer with water-seal bottom, large inclined grate, steam blast, poking and cleaning holes above the water level, and a bell and hopper feed. The capacity claimed is 12 tons in twenty-four hours; the producer is not stopped for cleaning, the ashes passing down the inclined grate. The advantages claimed for water-sealed producers are that, as the ashes are cold when taken out, all heat is returned to the producer in the form of steam, which also saves much live steam.

(9) *The Fraser-Talbot Producer.*[4] This producer consists of a cylindrical sheet of boiler iron riveted to four I-beam columns and lined with brickwork as shown. To the bottom of the shell is attached an inverted truncated cone of cast iron holding the ashes; its lower end is sealed by water held in the concrete ash pan. The blast pipe in the centre of the ash pan is of cast iron. Its cone-shaped roof holds the bearing for the water-cooled vertical shaft, carrying two water-cooled stirring arms. The shaft has a rotating and a vertical motion, both of which are obtained by gearing carried on a steel platform riveted to the tops of the I-beam columns. The drawing shows two coal hoppers closed by cones. The producer gasifies 2,500–3,000 pounds coal per hour and requires an engine of 3–4 horse power.

(10) *The Klepetko Producer* at Great Falls, Montana,[5] is doing efficient work for copper reverberatory smelting furnaces. Mechanical stirring of the producer saves not only labor, but the regular and continuous work furnishes a uniform gas, prevents the formation of clinkers, and greatly increases the capacity.

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[1] "Iron Age," May 19, 1898; New Form of Bell Coal Hopper: *op. cit.*, January 29, 1903.

[2] "Cassier's Magazine," August, 1902; "Fielden's Magazine," V, p. 550; "Stahl und Eisen," 1903, p. 1191.

[3] "Cassier's Magazine," IX, p. 53.

[4] "Iron Age," February 14, 1901.

[5] "Transactions American Institute of Mining Engineers," XXXIV.

(11) Other non-mechanical modern producers are those of Miller,[1] Morgan,[2] Forter.[3]

(12) *Producers for Wood, Sawdust, Peat, etc.*[4] Producers for light fuels must be spacious and high to hold a sufficient quantity of fuel and to prevent unconsumed air from passing through them; they must, further, be connected with condensers to free the gases from the large amounts of water vapor generated.

Examples: "Transactions American Institute of Mining Engineers," IX, p. 310.

(13) *Management.* A new producer has to be dried and warmed, like any other furnace, by a slow fire. In starting, a fire is made, the gases being allowed to pass off through the charging opening; fuel is added gradually, and the thickness of the bed is slowly increased. When well under way, the charging opening is closed and the gas main opened. In order to insure complete expulsion of air, which might cause explosions to take place, the bed is at first kept low so as to obtain gases containing CO_2 and water vapor, which drive out the air. In normal work, the gas in the producer ought always to be above atmospheric pressure, even when the producer is worked by natural draught, so as to prevent air from being sucked in. The regular operations are charging coal, poking it down, and removing ashes and clinkers. In order to insure uniform charging, Bildt[5] uses an automatic feed, which evenly distributes the coal and thus obtains greater uniformity in the quantity and quality of the gas. In order to ascertain the rate of sinking of the coal, iron rods are periodically inserted through the roof holes. They serve also to poke down the coal in case spongy hollow spaces form, and to break up any clinkers. The ashes and clinkers are removed periodically, the manner and frequency varying greatly with the different producers.

While the leading use of producer gas has been and remains to serve as a heating fuel, recently it has been used directly for the production of energy in a gas engine.

SECTION 71. **Distribution of Heat in Gas Producer and Open-hearth Furnace.** The subjoined table, taken from Campbell, "The Manufacture and Properties of Iron and Steel," New York, 1903, p. 232, is part of a summary of a detailed calculation by J. W. Richards, based upon the records of von Jüptner-Toldt,[6] and Campbell's Physical and Chemical Equations of the Open-hearth Process.[7]

The table shows, taking the Campbell-Richards data, that (1) the producer demands one-fifth of the heat value of the coal (21.6 per cent.) and delivers the remaining four-fifths (78.4 per cent.) as available combustible in the gas; (2) the combustion of the metalloids is one-half (12.4 per cent.) of the heat absorbed (24.7 per cent.) in heating and melting the charge; (3) the loss by radiation and combustion is one-half (49.1 per cent.) of all the heat supplied; (4) the loss through the waste gases is one-fourth (26.2 per cent.) of all the heat supplied; (5) heating and melting the charge utilizes only one-fifth (20 per cent.) of all the heat supplied.

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[1] "Iron Age," January 23, 1896.

[2] "Colliery Guardian," February 15, 1901.

[3] "Iron Age," June 13, 1901.

[4] Daniels, "Transactions American Institute of Mining Engineers," IX, p. 312; Odelstjerna, *op. cit.*, XXIV, p. 293; Merton, *op. cit.*, XXXIV; Loomis, "Journal Iron and Steel Institute," 1890, II, p. 280; Douglas, *op. cit.*, 1902, I, p. 287.

[5] "Transactions American Institute of Mining Engineers," XXVIII, p. 166; "Iron Age," April 30, 1896.

[6] "Chemisch-Calorische Studien über Generatoren und Martinöfen," Felix, Leipsic, 1888.

[7] "Transactions American Institute of Mining Engineers," XIX, p. 128; XXII, p. 395.

scrap, estimated to contain Si, 0.40; Mn, 0.90; C, 1.00, showed, after melting, Si, 0.02; Mn, 0.09; C, 0.54; the slag gave SiO_2, 50.24; MnO, 21.67; FeO, 23.91. The charge is introduced by hand, by a hydraulic or electric charging machine. Usually, first one-half of the pig iron is charged, then follows the scrap, and over this the rest of the pig iron is spread evenly. In this way the pig iron melting on the bottom will dissolve the scrap, and the pig iron on top, trickling over the hot scrap, will protect it from oxidation. There ought to be enough Si present to give the slag formed about 50 per cent. SiO_2; if not, sand may have to be added. With a charge of 25 per cent. pig iron with 1 per cent. Si and the adhering sand from the casting bed, the slag will have the required percentage of SiO_2.

In about four hours the charge will be melted down. After melting, the bath is kept hot to oxidize the remaining Si, Mn, and C; the oxidation of C is assisted by the addition of iron ore at short intervals. The CO set free by the reaction: $Fe_2O_3 + 3C = 2Fe + 3CO$, causes the metal to boil, and thus exposing new surfaces to the oxidizing action of the flame furthers the elimination of Si and Mn. Iron ore is added until the fracture of a sample cast into a small ingot (say 1 inch square) and chilled in water shows that the C has been eliminated as required. There is also enough time to make an approximate determination.

With a bath containing 1 per cent. C, 1,500 pounds of ore will reduce the C to 0.08 per cent.; the first 500 pounds will reduce it to 0.8 per cent., the second 500 to 0.4 per cent. To the metal bath above with 1 per cent. C were added 1,020 pounds hematite. The composition of the metal was changed to Si, 0.02; Mn, 0.04; C, 0.13; the slag to SiO_2, 49.40; MnO, 16.50; FeO, 29.79. If the temperature of the bath be kept very high, there may be a reduction of SiO_2 from the slag by means of C.

The next step is to add the necessary Mn to the steel in the form of spiegeleisen for high-carbon and of ferromanganese for low-carbon steel. The latter is usually heated and added to the ladle while tapping. The amount of Mn oxidized is less with a high-carbon than with a low-carbon steel; it decreases with the amount added. Thus, if 1.00 per cent. Mn be added to the bath, 0.60 per cent. will be found in the steel (loss 0.40 per cent.); if 0.60 per cent., the steel will retain 0.40 per cent. (loss 0.20 per cent.). If steel castings are to be made, Si must be added to prevent blowholes.[1] This is done by charging ferrosilicon (Si, 9.5 per cent.; Mn, 18 per cent.; C, 2.0 per cent.) to the bath in the furnace. Enough is added to give the steel Si, 0.25 per cent. It is assumed that 50 per cent. of the Si in the ferrosilicon is oxidized and enters the slag.

Tapping, casting, etc., are the same as in the basic open-hearth process (see later). After every charge, the hearth is patched with bottom mixture.

In the *Pig and Ore Process*, it is advisable to use pig iron running low in Si and Mn, as, if the pig iron runs high in these elements, the first additions of ore are consumed in removing them before they can act upon C. On account of the large mass of slag formed, it is not possible to hurry the operation. The process therefore is slow and expensive unless carried on in a tilting furnace, when the contents of the furnace, after most of the Si has been slagged, is emptied into a ladle and the metal then poured back with as little slag as possible.

The time required for a heat is from six to twelve hours, varying with the kind and size

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[1] Howe, "Steel," p. 41.

[illegible]

of the furnace, the manner of charging, and the amount of oreing. With direct metal, the time required is, of course, much reduced. The loss in iron is 5–8 per cent.[1]

SECTION 73. **The Basic Open-hearth Process.**[2] In the basic open-hearth process, beside Si, Mn, and C, practically all of the P and most of the S is removed. The elimination of P and S is made possible by the basic hearth, the addition of CaO, and the formation of a basic slag.

Pig, Scrap, and Ore Process. The proportion of pig iron and scrap is about the same as in the acid process. The pig iron ought to be lower in Si than for the acid process, in order to avoid the necessity of adding much flux to form a basic slag, which would increase the already large amount formed with basic pig iron. The percentage of P can be high, but the greater the amount of P the longer the time required to form a lime phosphate. As the S is only partly eliminated, its percentage ought to be low (under 0.1 per cent.).

The lime is usually added in the form of calcined limestone. The limestone ought to run low in SiO_2, so as to furnish as much available CaO as possible; it ought also to contain little MgO, as this has less power to hold P than CaO; it also makes the slags viscous. In charging, usually 5 per cent. burnt CaO is spread on the hearth and 5 per cent. mixed in with the charge; the rest is added later as required. Limestone may be used, but it retards the process by the heat absorbed in driving off the CO_2; this gas, however, bubbling up through the metal, stirs it, and also aids oxidation by giving off some O, being reduced to CO. In patching the bottom, limestone is used to fill the cavities, as it is not so readily floated up by the charge as burnt lime. The ore is mixed in with the charge. This can be done, as, being basic, it does not attack the bottom; further, being in intimate contact with the pig, it will act upon it while melting and have a stronger oxidizing effect. This is important as, assuming, *e.g.*, the Si, C, and Mn in an acid and basic charge to be the same, in the basic charge the P has to be oxidized in addition to the other elements. During the melting down, most of the Si,[3] Mn, and a large part of the C and P are removed. The boiling caused by the evolution of CO brings the metal into contact with air, which oxidizes the C and P; the latter is taken up by the basic slag. Lime is added at intervals should the slag not be sufficiently basic for this purpose. When the C has been reduced to 0.08 per cent., the P will have been eliminated down to 0.04 per cent. The fracture of a small chilled sample-ingot indicates the progress made in dephosphorization. With much P, it will show in the centre two crossed lines of brilliant crystals (phosphorus cross). An approximate analytical determination of P can be quickly made (measuring the yellow phospho-molybdate precipitate in a graduate tube). The sample will also be brittle. If the P has not been removed, the C is raised by adding pig to the charge (pigging back) and the metal bath again "boiled down." Thus the elimination of the P goes on with that of the C and is influenced by the ability of the slag to combine with the P_2O_5. This is shown in the subjoined table.[4]

---

[1] Ledebur, "Manual," 4th ed., p. 1049.

[2] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 282; "Transactions American Institute of Mining Engineers," XIX, p. 129; XX, p. 227; XXII, p. 345.

[3] "Journal Iron and Steel Institute," 1902, I, p. 54.

[4] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 286.

Group.	Pounds of ore charged with stock, per ton of metal.	Number of heats in group.	COMPOSITION OF METAL, PER CENT.						COMPOSITION OF SLAG AFTER MELTING, PER CENT.	
			P			C				
			Initial.	After melting.	Per cent eliminated.	Initial.	After melting.	Per cent eliminated.	SiO_2	FeO
1	None	17	0.20	0.046	77	2.00	0.71	65	19.21	13.68
2	None	4	1.36	0.594	57	1.50	0.60	60	14.90	n. d.
3	None	9	0.19	0.023	88	1.80	0.27	85	15.55	19.68
4	None	9	0.19	0.072	62	1.80	0.78	57	19.98	12.20
5	300	3	2.50	0.744	70	3.50	0.59	83	11.96	8.61
6	115	6	0.55	0.274	50	2.90	1.00	66	30.73	10.71
7	140	7	0.55	0.402	27	2.90	1.48	49	34.22	10.95

In groups 1–5, the elimination of P is satisfactory; in groups 6–7, it is imperfect, owing to the high percentage of SiO_2 (30–34 per cent.) of the slag.

In the following table, Campbell[1] represents an average of seventeen heats (one-half pig, one-half steel scrap, and no ore at start), containing C, 2.00 per cent.; Si, 0.40 per cent.; Mn, 0.85 per cent.; P, 0.20 per cent.

Test.	METAL.				SLAG.					
	Composition, per cent.				Composition, per cent.					
	C	Si	Mn	P	SiO_2	MnO	CaO	MgO	FeO	P_2O_5
A	0.71	0.06	0.33	0.046	19.21	11.12	42.16	6.64	13.68	5.149
B	0.34	0.01	0.25	0.022	16.37	10.36	42.78	7.87	16.29	4.848
C	0.12	0.01	0.22	0.013	15.08	9.01	42.16	8.45	20.34	3.850
D	0.16	0.01	0.49	0.018	15.75	14.11	39.05	10.40	16.65	2.961

Sample A was taken after complete fusion; sample B at the beginning of the boil after adding 1,965 pounds ore; sample C when the bath was ready for the recarburizer (775 pounds ore having been added between B and C); sample D after casting. The slags contain from 15 to 19 per cent. SiO_2 and 48 to 49 per cent. Ca(Mg)O. When little C is to be removed, SiO_2 may run up to 20 per cent.; when much P, it may have to go down to 12 per cent. If it contains over 55 per cent. Ca(Mg)O (see Saniter below), it becomes too viscous; if it contains under 10 per cent. SiO_2, it will not flow unless it is exceptionally rich in Fe, Mn, or P. Normal basic slags have the following composition:[2]—

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[1] "Manufacture and Properties of Iron and Steel," p. 285.

[2] Campbell, *op. cit.*, p. 293.

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[illegible]	[illegible]	[illegible]	[illegible]
[illegible]	[illegible]	[illegible]	[illegible]

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NORMAL BASIC OPEN-HEARTH SLAGS.

SiO_2	P_2O_5	FeO	$SiO_2 + P_2O_5$
20.72	6.36	16.20	27.08
19.04	8.24	20.16	27.28
12.40	13.73	12.60	26.13

The S of the metal bath is to a small extent removed by being directly oxidized; most of it, however, enters the slag as CaS, and this is in part decomposed by the strongly oxidizing atmosphere of the furnace. Campbell[1] has shown that as the SiO_2 increases in the slag, its capacity to absorb S decreases. Slags contain 0.02–0.50 per cent. S.[2] The Mn in the recarburizer has a strong desulphurizing effect, combining with the S to MnS; an addition of 0.60–0.75 per cent. Mn to the bath reduced the S to 0.01 per cent.[3] See also Massenez[4] and Ledebur.[5] Riemer[6] finds that the addition of manganese ore to the charge has a decided desulphurizing effect. Saniter[7] used $CaCl_2$ successfully to remove the S, but in order for it to be effective, the slag must contain 50–60 per cent. CaO, the $CaCl_2$ making the slag less viscous.

After the P has been taken out, the metal is recarburized as in the acid furnace. As the basic metal is liable to contain more Mn than the acid and therefore less O, less deoxidizer will be needed. Another point to be considered is the reduction of P from the slag and consequent rephosphorization of the steel. This is not so important in the basic open-hearth as in the basic Bessemer process, as the metal contains less O and therefore requires less deoxidation; further, ferromanganese being added in the solid state causes a less violent reaction than the liquid spiegeleisen of the converter. In order to recarburize the metal without adding Mn, pulverized coke or charcoal[8] is added in the ladle and is readily absorbed. According to Wedding,[9] the best way of applying the Darby Process is to mix lean coal with 7 per cent. burnt lime and enough water to permit molding into bricks. These are dried to expel the non-combined water and are then charged into the ladle. The heat of the metal slowly drives off the combined water; the bricks crumble and give off gradually their C, the ascending steam bubbles stirring the metal.

Whenever the hearth has to be repaired, it is patched by ramming in some burnt lime or some lime-tar mixture and bringing it to a set with the flame.

The following is a record made by A. L. Davis, February, 1899, at the Sharon Works of the National Steel Company. The furnace is called a 40-ton furnace, has upright regenerators underneath the furnace; it is heated with producer gas.

---

[1] "Transactions American Institute of Mining Engineers," XXII, p. 446.

[2] Ledebur, "Manual," 4th ed., p. 1048.

[3] Campbell, "Manufacture and Properties of Iron and Steel," p. 294.

[4] "Journal Iron and Steel Institute," 1891, II, p. 76.

[5] "Stahl und Eisen," 1896, p. 415.

[6] "Stahl und Eisen," 1902, p. 1357; "Iron Age," January 29, 1903.

[7] "Journal Iron and Steel Institute," 1892, II, pp. 216, 260; 1893, I, p. 73; report by Snelus, *op. cit.*, 1893, I, p. 73.

[8] *Darby Process:* "Transactions American Institute of Mining Engineers," XIX, p. 790.

[9] "Stahl und Eisen," 1894, pp. 473, 533; 1895, p. 570.

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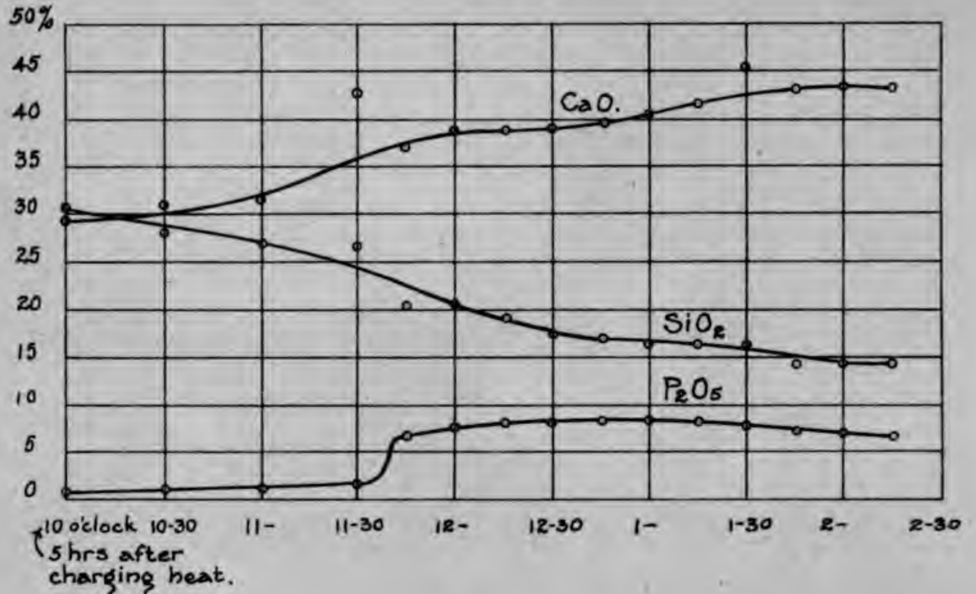
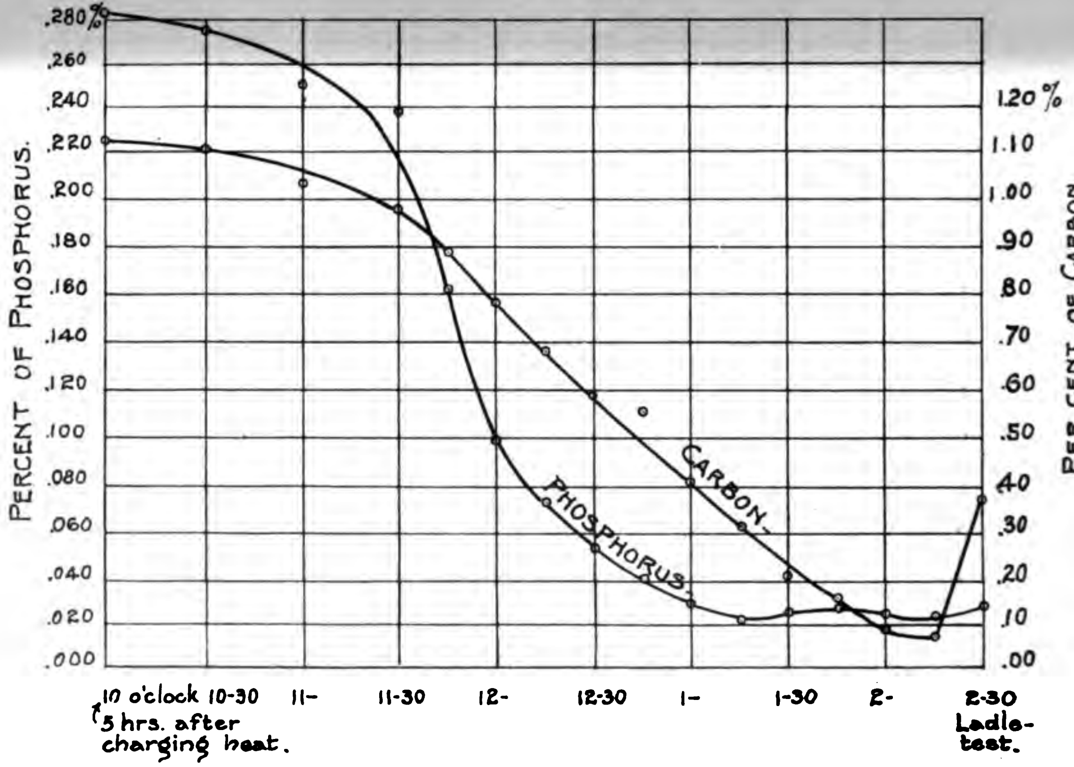


SLAG.

50%
45
40
35
30
25
20
15
10
5
0
CaO.
SiO2
P2O5
10 o'clock
5 hrs after charging heat.
10-30
11-
11-30
12-
12-30
1-
1-30
2-
2-30



METAL.

PERCENT OF PHOSPHORUS.
.280%
.260
.240
.220
.200
.180
.160
.140
.120
.100
.080
.060
.040
.020
.000
PER CENT OF CARBON.
1.20%
1.10
1.00
.90
.80
.70
.60
.50
.40
.30
.20
.10
.00
CARBON.
PHOSPHORUS.
10 o'clock
5 hrs. after charging heat.
10-30
11-
11-30
12-
12-30
1-
1-30
2-
2-30
Ladle-test.

History of Heat 1,041, February, 1899, Producing 17,900 Pounds Steel and 500 Pounds Scrap.

Charge, pounds.	Approximate Average Analysis of Metal Charged.				
	C	P	S	Mn	Si
Pig iron, 38,000	2.00	0.30	0.050	1.00	0.40
Scrap, 42,000					
Limestone, 6,000	...	...	...	...	...

Analyses of Materials.

	Limestone.	Dolomite.	Magnesia.	" Calcium."[1]	Ore.
SiO_2	0.98	1.93	5.42	0.28	2.06
Al_2O_3	1.26	1.50	0.78	26.61	1.10
Fe	0.28	0.60	5.60	4.56	67.50
CaO	54.01	31.72	9.75	60.29	1.40
MgO	0.60	20.50	75.05	1.19	1.50
P	0.006	0.01	...	...	0.134
MnO	...	...	...	0.37	0.50

[1] Calcium is the waste from the Pennsylvania Salt Manufacturing Company, Natrona, Pennsylvania, in treating cryolite

Sample cinder and steel.	Taken at	Remarks.
No. 1	10.00 A.M.	Charge not completely melted.
No. 2	10.30	Lime is starting to come up.
No. 3	11.00	Lime has about all come up.
No. 4	11.30	Started to work the heat and charged 470 lbs. ore.
No. 5	11.45	Charged 450 lbs. ore.
No. 6	12.00	Charged 300 lbs. ore.
No. 7	12.15 P.M.	Charged 225 lbs. "calcium."
No. 8	12.30	Charged 320 lbs. ore.
No. 9	12.45 P.M.	Charged 500 lbs. ore.
No. 10	1.00	Charged 250 lbs. ore, 145 lbs. "calcium."
No. 11	1.15	Charged 200 lbs. ore.
No. 12	1.30	Charged 240 lbs. ore.
No. 13	1.45	Charged 200 lbs. ore.
No. 14	2.00	
No. 15	2.15	

Three hundred pounds ferromanganese, 125 pounds ferrosilicon, and 375 pounds coal were added in the ladle. For repairing bottom, 800 pounds of dolomite and 750 pounds of magnesite were used.

ANALYSES OF STEEL.

No.	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Ladle.
C	1.12	1.11	1.04	0.98	0.89	0.78	0.68	0.58	0.56	0.41	0.32	0.21	0.16	0.09	0.07	0.37
P	0.282	0.275	0.250	0.238	0.163	0.100	0.074	0.055	0.042	0.030	0.024	0.026	0.028	0.025	0.024	0.028
S	0.050	0.043	0.049	0.042	0.043	0.043	0.040	0.032	0.044	0.040	0.032	0.032	0.034	0.038	0.033	0.030
Mn	0.55	0.53	0.54	0.50	0.36	0.35	0.31	0.32	0.37	0.40	0.31	0.30	0.31	0.28	0 27	0.38
Si	0.015	0.013	0.009	0.010	0.009	0.008	0.008	0.009	Trace	Trace	Trace	Trace	Trace	Trace	Trace	0.01 av.

ANALYSES OF CINDERS.

SiO_2	30.20	28.00	27.10	26.76	20.40	20.68	19.21	17.70	17.26	16.40	16.36	16.32	14.38	14.30	14.28
Al_2O_3	0.78	1.09	1.67	0.94	0.76	0.79	1.09	1.79	1.58	2.66	2.57	1.86	0.84	0.91	0.88
FeO	5.12	12.72	7.74	5.64	15.08	12.72	10.36	12.33	11.55	13.68	16.10	13.77	16.71	17.49	16.97
CaO	29.64	30.66	31.62	42.87	36.89	38.86	38.96	38.99	39.68	40.65	41.55	45.27	43.29	43.26	43.24
MgO	14.48	8.78	16.62	7.91	6.36	5.36	8.83	8.22	7.96	6.19	4.56	4.93	6.70	7.13	7.35
MnO	18.39	16.84	13.30	13.12	12.99	12.97	12.91	12.86	12.18	11.91	11.21	10.07	10.25	9.80	9.80
P_2O_5	0.788	1.07	1.19	1.84	6.78	7.57	7.93	8.25	8.36	8.29	8.16	7.86	7.19	6.93	6.83
P	0.333	0.452	5.03	0.777	2.86	3.20	3.49	3.49	3.53	3.50	3.45	3.32	3.04	2.93	2.89
Si	14.93	13.05	12.60	12.49	9.52	9.65	8.96	8.26	9.15	7.65	7.63	7.62	6.71	6.67	6.67
Mn	13.21	13.09	9.55	9.42	9.33	9.33	9.27	9.24	8.75	8.55	8.05	7.23	7.36	7.04	7.04
Fe	3.98	9.89	6.02	4.38	11.73	9.89	8.06	9.59	8.98	10.64	12.52	10.71	13.00	13.65	13.20

WEIGHT OF CINDERS.

Total weight, lbs.	3,955	3,767	5,525	4,788	4,662	5,000	5,400	5,600	5,807	6,171	6,406	6,596	7,171	7,508	7,613
Weight in lbs. per ton metal charged	98	90	140	119	116	125	135	140	145	154	160	165	179	185	190

The elimination of C and P from the metal and the percentages of SiO_2 and $CaO + MgO$ given in the above tables are graphically represented in Figures 271–272.

In tapping a stationary acid or basic furnace, a bar is driven through the tap hole and the whole content run out into a ladle similar to the one used with the Bessemer converter. With tilting furnaces, the sand closing the spout is removed and the content of the furnace poured into the ladle; *i. e.*, the furnace is tilted sufficiently to have the metal run out completely before any slag can reach the spout. From the ladle the steel is teemed either as in the Bessemer process or by bottom-casting, as shown in Figure 273. *F* is a cast-iron square or circular bottom plate large enough for from four to eight molds, *A*. It contains a number of bottom runners

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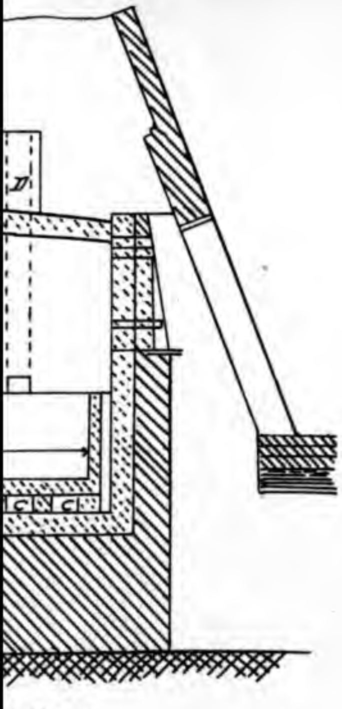

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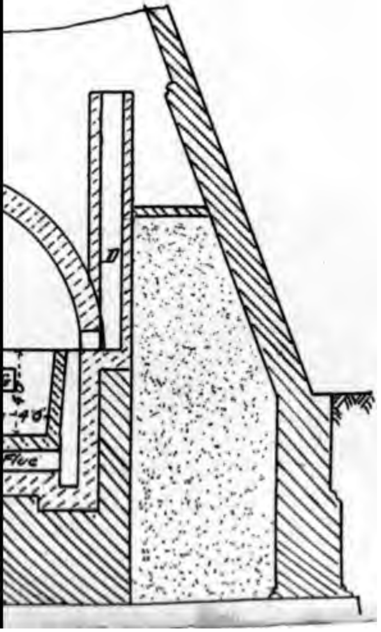
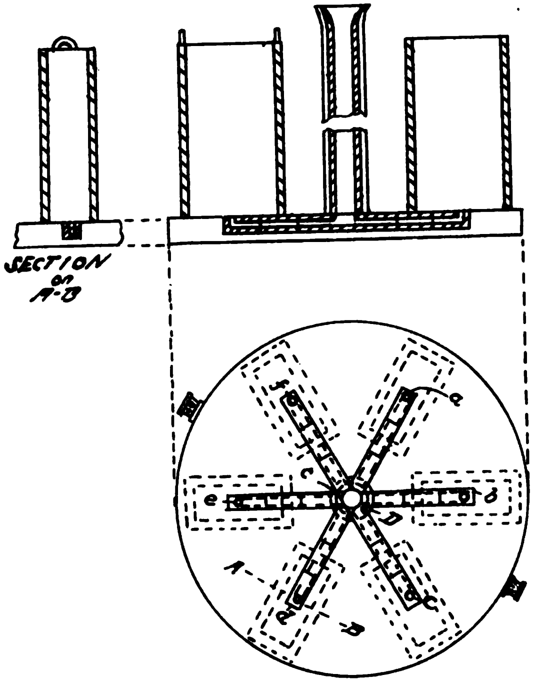
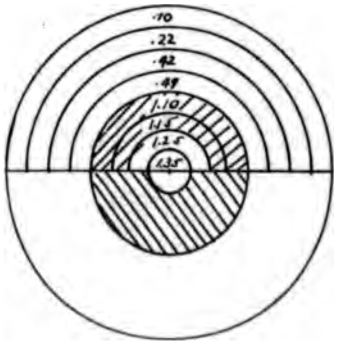


FIG 273



BOTTOM CASTING

FIG 276



DIFFUSION of CARBON.

lined with hollow brick, *H*. The bottom plate is covered with movable cast-iron plates, *G*. Over each bottom runner is placed a warmed mold, *A*, smoked or whitewashed on the inside; over the centre the rising head *D* with funnel *C*, both of which are lined with brick *H'*. After each cast, the steel adhering to the ingots is knocked off and goes back to the next charge; the head and bottom are relined with brick, etc. Bottom-casting has the advantage that there is no spattering on the sides of the mold and that the occluded gas is given off more readily than in top casting. It is practiced especially when the ingot must be absolutely free from defects, *e. g.*, if it is to be rolled into plate. Sometimes very large ingots are cast to be reduced in a "slabbing mill" and then cut into slabs of exact weight. These, after having been inspected and having defects chipped out, are rolled in a plate mill into plates to be used for boilers or other purposes. If a slabbing mill is not in use, flat ingots are cast as near as possible to the required weight, and are then "rolled direct."

The basic process takes less time for a heat than the acid process, but, making more slag, the furnace produces less steel per square foot of hearth area. The waste in iron is about the same in both processes. On account of the greater cheapness of basic pig iron in this country, basic open-hearth work is more common than acid, although the cost of the burnt lime may to some extent affect the higher price of the acid pig iron.[1] A comparison of acid and basic open-hearth steel is given below.

SECTION 74. **Cost of the Open-hearth Steel Process.** The following is an estimate of cost by C. D. Wright for 1889:[2]—

ELEMENTS OF COST IN 1 TON (2,240 POUNDS) ACID OPEN-HEARTH STEEL INGOT.

Locality.	Materials.							Labor.	Officials and clerks.	Fuel.	Supplies and repairs.	Taxes.	Total.
	Pig iron.	Scrap.	Ferromanganese.	Other.	Total.	Value of scrap, etc.	Net.						
United States . .	$2.813	$15.375	$0.650	$3.375 a	$22.213	$1.700	$20.513	$3.000	$0.250	$1.90	$1.700	$0.025	$27.388
Continent of Europe	13.093	Cost of all materials included in pig iron.			13.093	0.113	12.980	0.976	0.108	1.124	1.563	0.015	16.766
Great Britain . . .	8.275	4.251	0.415	0.670 b	13.611	0.373	13.238	1.589	0.068	0.880	1.083	0.009	16.867

a = extra puddled muck-bar iron. b = iron ore.

---

[1] Tordeur, "Revue Universelle des Mines," 1892, XX, p. 1; "Iron Age," December 21, 28, 1893; "Stahl und Eisen," 1893, p. 101.

[2] Sixth Annual Report of Commissioner of Labor, 1891, p. 156.

Campbell[1] gives the following estimate of cost for the Pittsburgh district:—

ACID OPEN HEARTH.

Six tons pig iron	@ $11.00	$66.00
Eighteen tons scrap	@ 12.00	216.00
One-third ton best ore	@ 3.75	1.25
Total		$283.25

Product, 23 tons of steel.

Cost of stock, $12.32 for each ton of finished product.

BASIC OPEN HEARTH.

Six tons pig iron	@ $10.00	$60.00
Eighteen tons scrap	@ 12.00	216.00
One-third ton best ore	@ 3.75	1.25
3,000 pounds pure burnt lime		6.00
Total		$283.25

Product, 23 tons steel.

Cost of stock, $12.32 for each ton of finished product.

For labor, $6 per ton of steel may be added. See also edition of 1903 of Campbell's book, page 335. The market values of Bessemer and open-hearth steel above the cost of pig were given in Section 46.

SECTION 75. **Modifications of the Open-hearth Process.**

(1) *The Bertrand-Thiel Process.*[2] In this process, which is a modification of the basic open-hearth process, first, pig iron is melted down in one (the primary or rough refining) furnace and the elimination of phosphorus and silicon started. With direct metal, first some ore is charged, then scrap pig iron, and lastly the direct metal run in. The amount of ore is so regulated that the melted bath be practically free from Si and retain about 0.2 per cent. P and 2.5 per cent. C. Second, scrap iron and ore are melted down at the same time in a second (the secondary or finishing) furnace situated at a lower (10 + feet) level, where the iron becomes somewhat oxidized. Third, the metal, rough-refined in about three hours, is tapped from the upper into the lower furnace and its slag diverted during the flow. The two charges reacting quickly, the C and P are rapidly removed. By the removal of S and the partial elimination of C and P in the upper furnace with a slag that is not very basic, relatively little lime has to be added. The result is that a smaller quantity of slag per unit of iron is produced than in the regular practice, which aims to remove the whole of the phosphorus in one operation. This smaller amount of slag leaves more room in the furnace for the metal charge. The partly treated metal charge free from slag meets in the lower furnace iron oxide containing very little slag, hence the reaction is vigorous and the amount of slag formed small. This again leaves more room for the metal. The higher temperature over that in the ordinary process is explained by the higher initial temperature of the substances brought together. The frothing of the charge in the lower furnace during the boiling period is less, on account of the smaller amount of slag and its higher temperature. The slag from the upper furnace contains as much P as that of the basic Bessemer converter, viz., 16–20 per cent. P_2O_5, and has commercial value. Basic open-hearth slags contain 1–3 per cent. P_2O_5.

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[1] "Properties and Manufacture of Structural Steel," New York, 1896, 1st ed., p. 164.

[2] "Iron Age," April 8, 1897, Gilchrist; May 2, 1901, Cabot; August 7, 1902, Christie; April 2, 1903, Thiel. "Transactions American Institute of Mining Engineers," XXVI, p. 380; XXVIII, p. 254; XXX, p. 531, Hartshorne. "Engineering and Mining Journal," September 2, 1899, Howe. "Stahl und Eisen," 1897, pp. 403, 733; 1898, p. 146; 1901, p. 305; 1903, p. 306, Thiel; 1903, p. 36, Ledebur. "Journal Iron and Steel Institute," 1897, I, p. 115, Bertrand. Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 315.

(2) *The Talbot Continuous Process.*[1] This consists, first, in adding to an initial bath of steel (forming 60–75 per cent. of the capacity of the furnace, which is a basic 80-ton Wellman tilting furnace, 30 x 10 feet, with central gas and lateral air flues) charges of molten pig iron and of mill cinder and iron ore to enrich the slag; and second, in withdrawing aliquot part of steel and slag, the oxidizing power of which has been exhausted. The process is based upon the desiliconizing, decarburizing, and dephosphorizing effect of a highly basic ferruginous slag upon molten pig iron. While the melting pig iron passes through the slag, and while it remains in contact with it (floating upon the heavier steel), Si, C, and P are gradually removed. The process is adapted for melted pig iron alone; scrap pig iron must be melted first, say in a cupola. From a furnace of 70 tons capacity, steel casts of from 20 to 30 tons are made. The weekly output is about 650 tons steel. With a pig iron containing C, 3.7 per cent.; Si, 1 per cent.; S, 0.060 per cent.; P, 0.90 per cent.; Mn, 0.40 per cent., there were required to produce the above 650 tons steel 4 tons iron ore, 5 tons manganese ore, 60 tons cinder and scale, 50 tons limestone, and $2\frac{1}{2}$ tons ferromanganese and silicospiegel. The average yield of steel was 106 per cent. or 6 per cent. more than the weight of the pig metal charged. Surzycki[2] has adapted the Talbot process to the stationary open-hearth furnace by supplying it with two or more tap holes, one diagonally above the other and leading into a double spout through which the whole or only part of the contents of the furnace can be tapped.

The Morrell Process.[3] In this process, limestone and iron ore are charged onto the hearth of a stationary basic open-hearth furnace and heated until pasty or nearly fused, then molten pig iron is poured onto it, when a rapid reaction occurs which frees the metal from Si and Mn and reduces the C one-half. The slag formed is tapped off and the metal converted into steel in the usual way by the addition of iron ore and lime. The blast-furnace metal, *e.g.*, contained C, 4.00 per cent.; P, 0.065 per cent.; Si, 0.70 per cent.; Mn, 0.85 per cent.; S, 0.06 per cent.; the reaction changed these figures to C, 2.25 per cent.; P, 0.04 per cent.; Si, Mn, S, 0.04 per cent. A stationary furnace of 40 tons capacity makes seventeen heats (at eight to nine hours) per week, producing 690 tons steel. The preliminary charge is 33.5 tons limestone and 12 tons hematite (or its equivalent of mill scale); it is heated for one and one-half hours; 44 tons of direct pig metal are poured in; the reaction occurs at once and the impure slag is withdrawn within one hour; the yield is 102 per cent. of the weight of the pig iron charged.

Electric Production of Steel from Pig Iron. This is discussed in Sections 84 and 85.

SECTION 76. **Comparison of Bessemer, Acid, and Basic Open-hearth Steel.** Campbell,[4] in comparing Bessemer and open-hearth steel (acid and basic) with 0.06–0.10 per cent. P, gives open-hearth steel decidedly the preference —

(1) As being more reliable. In comparing Bessemer and open-hearth steel of the same chemical composition by the ordinary methods of testing, no difference will be noticed; but Bessemer metal has been found to suffer a larger number of inexplicable breakages. Microscopical examinations may furnish the clew needed.

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[1] "Iron Age," February 8, 1900, Talbot; February 8, 1900, Christie. "Journal Iron and Steel Institute," 1900, I, p. 33; 1903, I, p. 57, Talbot. "Stahl und Eisen," 1900, p. 263; 1903, p. 170, Talbot. Campbell, *op. cit.*, pp. 310, 326, 330. Patent decision in favor of Morrell: "Engineering and Mining Journal," February 11, 1904.

[2] "Stahl und Eisen," 1904, p. 163; "Iron and Steel Magazine," I, p. 492.

[3] "Journal Iron and Steel Institute," 1900, I, p. 71; "Iron Age," August 7, 1902, p. 23.

[4] "Transactions American Institute of Mining Engineers," XXII, p. 350; "Manufacture and Properties of Iron and Steel," 1903, pp. 15, 529.

(2) As being more homogeneous. Test pieces taken from different parts of a Bessemer steel plate have shown differences in their mechanical properties, which is not the case with open-hearth plate.

(3) As being more uniform; because the steel can be and is tested chemically and mechanically before it is tapped. This gives a better control of the operation than in the Bessemer process.

Ledebur,[1] on the other hand, representing especially German experience, says that while experiments made in the '70's and '80's seemed to justify the belief that open-hearth metal was preferable to Bessemer metal, the more recent investigations[2] have proved that there is no reason to prefer one to the other for structural use.

There is no difference in the mechanical behavior of acid and basic steel, although a prejudice exists in favor of the acid steel, starting, as it does, with purer material. Basic steel will, however, be used more and more, as the P limit is being constantly reduced. Formerly the limit was 0.10 per cent.; at present it is 0.08 per cent.; it will soon be 0.06 per cent. and, according to Campbell,[3] it ought to be 0.04 per cent. It is very difficult to obtain pig and scrap giving in an acid furnace a metal bath with 0.04 per cent. P. The difficulty has been overcome at the works of the Pennsylvania Steel Company, Steelton, Pennsylvania,[4] by treating the pig iron in a basic furnace and transferring and finishing the partly refined metal in an acid furnace.

SECTION 77. **Open-hearth Steels.**

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[1] "Manual," 4th ed., p. 1052.

[2] "Stahl und Eisen," 1891, p. 707; 1892, pp. 279, 558, 593; 1893, p. 275.

[3] "Transactions American Institute of Mining Engineers," XXII, p. 353.

[4] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 302.

Boiler-plate steel must contain < 0.35 per cent. P, run low in S and Mn, and have enough C to keep the tensile strength above 50,000 pounds. Bridge steel usually contains 0.08 per cent. P, but on account of the liability of high-P steel to break under sudden shock, a more prudent limit is 0.05 per cent. P. Steel castings are better if made low in P; the desired hardness is obtained from carbon.

A list of articles from open-hearth steel, arranged according to their carbon contents, is published in the "Iron Age," July 17, 1902, p. 4. The American Standard Specifications and Methods of Testing Iron and Steel, by A. L. Colby, are reprinted in Campbell's "Manufacture and Properties of Iron and Steel," 1903, pp. 548–600.

E. STEEL CASTING.[1]

SECTION 78. **Steel Casting.** In 1902[2] there were in operation in this country eighty-five steel-casting plants producing a little over 300,000 tons cast steel.

The casting of steel resembles in many of its features ordinary foundry work, so that it is necessary to discuss only the leading differences. These appertain to the furnace, the metal, the mold, and the finishing (annealing) of the casting. Steel castings are made from open-hearth furnaces[3] with plants having a capacity of over 15 tons, and from baby Bessemer converters when the capacity is less than 15 tons; the former will be used with large, the latter with small castings. Open-hearth furnaces for making castings do not hold more than 20 tons metal; a capacity of less than 10 tons is not uncommon. Of the eighty-five steel foundries in operation in 1902, fifty-seven used the open-hearth furnace, eleven the (baby) Bessemer converter, ten the crucible furnace, and seven other apparatus not specified. The metal ought not to contain[4] over 1.20 per cent. Mn, 1.60 per cent. Si, 0.09 per cent. P, 0.08 per cent. S. American Standard Specifications (see above) require that ordinary castings shall not contain over 0.40 per cent. C nor over 0.08 per cent. P, and that castings to be tested shall not contain over 0.05 per cent. P nor over 0.05 per cent. S. The usual range of Si is 0.20–0.45, average 0.10 per cent. in mild, 0.30 per cent. in hard steels; of Mn 0.50–1.00, average 0.7 per cent. in soft, 0.80 per cent in hard steels; of C 0.2–0.6, average 0.35; the larger the casting the lower the C; with 0.15 per cent. C the steel is tough; with 0.70 per cent. C it is so hard that machining is slow. The effect of Si and Mn and the addition of less than 0.1 per cent. Al on the soundness of the casting will be discussed under heading G.

The soundness of a casting depends also upon the pouring temperature (see Section 56, Ingot Steel). The steel having been brought to a "dead melt" (*i.e.*, the chemical reactions have taken place and the metal is covered with a fluid slag that has no chemical effect upon the metal), the casting temperature will depend mainly upon the size of the casting, a large and simple casting being poured at a lower temperature than one that is small and intricate. The lower the temperature at which the metal is cast the better. Too low a temperature is liable

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[1] Harbord, "Steel," p. 212; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 591; Pourcel, Manufacture: "Journal Iron and Steel Institute," 1882, II, p. 509; Arnold, Properties: *op. cit.*, 1901, I, p. 175; Specifications, preceding paragraph; Sexton, Annealing: "Transactions American Institute of Mining Engineers," XXXV. Steel Foundries: Pennsylvania Steel Company, "Foundry," 1902, XX, p. 136; Shickle, Harrison, and Howard, East St. Louis, Illinois, "Iron Trade Review"; other foundries, see page 173.

[2] "Iron Trade Review," February 26, 1903.

[3] Campbell, Tilting Steel-casting Furnace: "Foundry," 1902, XX, p. 137.

[4] Mahler, "Génie Civil," XVIII, No. 12, through Ledebur, "Eisen- und Stahl- Giesserei," 1892, p. 380.

to give a defective surface; too high a temperature is sure to make an unsound casting, on account of the high shrinkage, which amounts to 1.5–2 per cent., or about $\frac{3}{16}$ inch to the foot.

The molds are of green and dry sand (see Section 40). Green sand will be used and top casting practiced with simple and light, dry sand and bottom casting with medium and heavy castings.

Finishing, *i. e.*, annealing, of steel castings becomes necessary, on account of the internal stresses caused by the large amount of shrinkage during solidification and cooling, and by the coarse crystallization, due to cooling down from a high temperature without any work put upon it. The annealing temperature is about 900° C. for low-C castings and 800–850° C. for castings with 0.5 + per cent. C. The time required will be the greater the larger the castings. The following experiments by Carr[1] bring out clearly the effects of annealing: —

STEEL CASTING: C, 0.27; Mn, 0.85; Si, 0.35; S, 0.020; P, 0.025.

Treatment.	Tensile strength, pounds per square inch.	Elongation, per cent.	Reduction of area, per cent.
Not annealed	84,645	11.7	12.2
	71,230	15.6	13.3
	88,126	19.5	23.1
	77,540	6.25	. . .
Properly annealed	92,465	24.2	38.3
	78,506	27.4	32.5
	69,595	32.0	42.2
	74,505	25.0	48.8
Improperly annealed	71,830	11.0	14.6
	66,803	10.9	11.4
	86,304	18.8	22.0
	97,714	18.7	13.4

AMERICAN STANDARD SPECIFICATIONS FOR STEEL CASTINGS: PHYSICAL PROPERTIES.

Test.	Hard castings.	Medium castings.	Soft castings.
Tensile strength, pounds per square inch	85,000	70,000	60,000
Yield point, pounds per square inch	38,250	31,500	27,000
Elongation, per cent. in 2 inches	15	18	22
Contraction of area, per cent.	20	25	30

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[1] "Foundry," 1902, XXI, p. 38; "Metallographist," 1903, VI, p. 316.

F. THE CEMENTATION PROCESS.

Literature: Percy, "Iron and Steel," p. 768; Harbord, "Metallurgy of Steel," p. 220; Flather, "Journal Iron and Steel Institute," 1902, I, p. 578.

SECTION 79. **Outline.** In the Bessemer and open-hearth processes of making steel the raw material was pig iron from which the C was removed. In the cementation process the start is made with wrought iron into which the necessary C is introduced. If wrought-iron is embedded in charcoal and brought to a yellow heat it will take up C and be converted into blister or cement steel. If the blister steel is cut and piled, heated and rolled, *shear steel* is obtained; if the operation is repeated on shear steel the result is *double shear steel;* if the blister steel is melted down in a crucible to free it from the small particles of slag contained in the original wrought iron, one form of crucible steel is produced. The process is probably 200 years old, and is only used now to a limited extent, as it is slow and expensive; but employing only the purest raw material, it furnishes an excellent finished product, especially after melting in the crucible.

SECTION 80. **Details.** *The Raw Materials* required in the process are wrought iron and charcoal. The wrought iron used is of the purest kind, containing under 0.02 per cent. P, and being practically free from Mn, Si, and S. It is used in the form of bars 2½–3 inches wide by ⅝–¾ inch thick by 10–18 feet long, somewhat shorter than the converting chests. The charcoal is pulverized to a size ranging from ¼ to ¾ inch and is freed from dust; that from deciduous trees is said to be most effective.

The Furnace[1] is shown in Figures 274–275.[2] There are two converting chests, *A*, heated from a fireplace, *B*, the flame passing underneath the chests through flues *C* and rising in the smaller flues on the outer sides and the larger central one between the chests. In the front is a trial hole, *G*, through which bars are introduced to examine the progress of the operation; *D* are short chimneys communicating with the conical hood *E*; *F* = door for charging and discharging; *H* = charging holes. A chest 30–48 inches wide by 30–60 inches deep and 10–18 feet long is made of sandstone, fire brick, or cast iron.

The Mode of Operating is as follows: On the bottom is placed fine sand or ground brick to close up any cracks that may form and exclude the air; then follows a bed of 3 inches of charcoal. Upon this is placed, flat side downwards, a layer of bars nearly touching one another; then comes a ½-inch layer of charcoal, to be followed by a second layer of bars, and so on until the chest is nearly filled, holding 8–18 tons of bars. Near the top layer of bars are inserted the trial bars. The whole is covered with grindstone waste or clay. The charging door and charging holes are bricked up and the fire started. In one and one-half to three days a yellow heat is reached and then maintained for seven to eleven days, according to the grade of steel desired (seven to eight for mild, nine and one-half for medium, eleven for high-C heats), the progress of the cementation being known by examining the trial bars. Cooling lasts about nine to fourteen days. The Crescent Steel Works, Pittsburgh, Pennsylvania, have a converting chest 4 x 5 feet and 18 feet charged with 50 tons of bars; heating up lasts one week, full fire one week, and cooling one week.

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[1] See also Bildt, Cementation with Gas: "Iron Age," October 3, 1901; "Stahl und Eisen," 1902, p. 438.

[2] Harbord, "Steel," p. 220.

The Product. The Cement Bars are covered more or less with blisters $\frac{1}{16}$–$\frac{3}{4}$ inch in diameter, resulting from CO[1] formed by the action of C upon the Fe_xO_y of the slag contained in the wrought iron. As in firing up a furnace one part is liable to be heated up more quickly than another, and therefore the hotter part remain a longer time at a full heat than the cooler, the degree of cementation of the bars will vary. They are broken and graded by fracture according to their percentage of C into six or seven classes. The saturation point lies at about 1.5 per cent. C. A bar when overheated and slightly fused becomes "glazed"; if air has found access to the pot a bar becomes oxidized, "aired." The microstructure of cementation steel has been studied by Saniter[2] and Arnold.[3] In the process two-thirds of the charcoal has been converted into a soot-like dust which is screened off and replaced by fresh charcoal. For a ton of steel 60 pounds of charcoal and 1,600–2,000 pounds coal are required.

Case Hardening[4] is a special form of cementation by means of which articles forged from wrought iron or mild steel are hardened that they may resist mechanical wear while retaining a tough ductile core, and perhaps take a satisfactory polish.

Harveyizing[5] of armor plate[6] is a cementation process. Illuminating gas conducted over a hot armor plate has a cementation effect on account of the acetylene (C_2H_2) it contains.[7]

The Chemistry of the Process. Cementation is based on the diffusion of solid C. It begins at 750° C.[8] and increases with the temperature and the length of exposure.[9] Arnold and Williams (*l. c.*) proved the process to be one of diffusion by placing in a cylinder of nearly carbonless iron (0.05 per cent. C) a tight-fitting core of steel (1.78 per cent. C), as shown in Figure 276 where the shaded portion represents the steel cylinder, and heating in *vacuo* for ten hours to 1,000° C. They analyzed the annular layers for C as they were turned off in a lathe. The percentages of C in the concentric rings show how the 1.78 per cent. C of the steel core migrated outward, leaving in the centre 1.35 per cent. C, and increasing the C in the iron at the periphery from 0.05 to 0.10 per cent. The rate at which the C penetrates the metal is about $\frac{1}{8}$ inch every twenty-four hours.[10] It is not uniform, but decreases as the saturation point

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[1] Percy, "Journal Iron and Steel Institute," 1877, p. 460.

[2] "Journal Iron and Steel Institute," 1896, I, p. 122.

[3] *Op. cit.*, 1898, II, p. 185; "Metallographist," II, p. 16.

[4] Woodworth, "Hardening, Tempering, Annealing, and Forging of Steel," New York, 1903, p. 129. Thallner-Brannt, "Tool Steel," Philadelphia, 1902, p. 119. "Iron Age," October 5, 1899; "Iron Trade Review," November 2, 1899, January 26, 1902, September 17, 1903; "Journal Iron and Steel Institute," 1903, II, p. 678.

[5] Harbord, "Steel," p. 237; "Stahl und Eisen," 1892, p. 213; "Iron Age," June 15, 1893, January 6, 1898. Patents, August 29, October 31, 1901.

[6] Harbord, "Steel," p. 237; "Stahl und Eisen," 1885, pp. 62, 131 184 (Brink); 1894, p. 552 (Kupelwieser); 1892, pp. 209, 454; 1893, p. 137; 1895, pp. 12, 793, 841; also "Iron Age," February 20, 1896; 1898, p. 894; 1902, p. 940 (Castner). Development of Armor Plate: "Iron," XL, p. 28 (Garrison); "Iron Age," May 28, 1903, p. 40 (S. G. H.); October 22, 1896 (Jaques). Present Status: *op. cit.*, December 6, 1894 (Sampson). Manufacture: *op. cit.*, May 30, 1895, (Ackermann). American and European Compared: *op. cit.*, February 10, 1898 (Neill). Krupp Process: *op. cit.*, March 2, May 25, 1889; Krupp Plate: *op. cit.*, July 10, 1902. Vickers Works: "Engineering," 1897, LXIV, pp. 555, 584, 607. Tests, Bethlehem Plate: "Iron Age," May 20, 1897; Bethlehem-Krupp Plate: *op. cit.*, November 10, p. 24, 1898; Carnegie-Krupp Plate: *op. cit.*, August 18, September 1, 19, November 24, 1898. New Process: *op. cit.*, August 7, 14, 1902.

[7] Castner, "Stahl und Eisen," 1895, p. 842.

[8] Arnold and Williams, "Journal Iron and Steel Institute," 1899, I, p. 85.

[9] Mannesmann, "Verhandlungen des Vereins zur Beförderung des Gewerbefleisses," 1879, p. 31.

[10] See also Roberts-Austen, "Journal Iron and Steel Institute," 1896, I, p. 139.

is reached.[1] The behavior of S, P, and Si has been studied by Boussignault,[2] that of Si more recently by Stead.[3]

The following analyses of cement steel give the two forms in which C is commonly present: —

Fracture.	Total carbon.	Carbide carbon.	Hardening carbon.	Temper carbon.	Reference.
Very coarsely crystalline	1.48	0.82	0.66	. . .	Thallner, "Stahl und Eisen," 1899, p. 914.
Coarsely crystalline	1.42	0.82	0.60	. . .	
Finely crystalline with some coarse particles .	1.51	0.765	0.745	. . .	
Finely crystalline	1.31	0.63	0.68	. . .	
Very finely crystalline	1.01	0.47	0.54	. . .	
. . . .	1.20	0.97	0.19	0.04	Ledebur, "Manual," 4th ed., p. 1085.

In England, where cementation is very prominent, the heats are named according to the amount of conversion.[4]

Spring heat,	0.50 per cent. C,	large iron centre, small steel rind.
Country heat,	0.63 per cent. C,	smaller iron centre, larger steel rind.
Single shear heat,	0.75 per cent. C,	iron centre and steel rind about equal.
Double shear heat,	1.00 per cent. C,	iron centre smaller than steel rind.
Steel through heat,	1.25 per cent. C,	iron centre gone.
Melting heat,	1.50 per cent. C,	iron centre gone.

G. THE CRUCIBLE PROCESS.

Literature: Howe, "Steel," p. 296; Harbord, "Steel," p. 228; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 148.

SECTION 81. **Outline.** The crucible steel process consists in melting down in a crucible suitable iron with or without carburizing or oxidizing agents, in tranquilizing (killing) the steel so as to obtain a compact casting, and in casting (teeming) it into ingots.

According to the composition of the charge, the following methods of working may be distinguished: —

(1) *Huntsman Method* (the original method, date about 1770[5]). Blister, or other highly carburated steel, is melted down alone or with glass, the aim being to render the steel homogeneous by removing particles of slag contained in the raw material. Bessemer and open-hearth

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[1] Saniter, "Journal Iron and Steel Institute," 1897, II, p. 121; see also Charpy, "Comptes Rendus," CCXXXVI, p. 1000, and Guillet, *op. cit.*, p. 1319.

[2] "Annales de Chimie et de Physique," ser. 5, V, p. 175.

[3] "Journal Iron and Steel Institute," 1903, I, p. 271.

[4] See Seebohm, "Journal Iron and Steel Institute," 1884, II, p. 379.

[5] Hadfield, "Journal Iron and Steel Institute," 1894, II, p. 224; Beck, "Geschichte des Eisens," III, p. 271.

steel is occasionally poured into crucibles and kept molten in a Siemens crucible furnace for half an hour, in order to remove oxides and gases, and thus make a stronger metal.

(2) *Heath's Method.* Manganese oxide and charcoal are added to the Huntsman charge; some manganese is reduced and improves the quality of the steel. This method is said to be the one prevalent in Sheffield, England.

(3) *Carburizing Fusion.* Bar iron or puddled bar is melted, with the addition of charcoal or washed metal (about 2 per cent. C). This is the common method in this country.

(4) *Uchatius (Swedish) Method.* Granulated pig iron and iron ore form the charge.

(5) *Pig and Scrap Method.* Pig iron and wrought iron or steel scrap form the charge.

SECTION 82. **Details of Carburizing Fusion.** *The Raw Materials* used in this country are puddled and bloomery iron, wrought iron, blister steel, and steel scrap. The additions, beside charcoal and washed metal, are spiegeleisen and ferromanganese[1] to prevent blowholes, to promote forgableness, to increase the absorption of Si and C, and to thin the slag. Salt is added to thin the slag, and potassium ferrocyanide to assist in carburization. It is common practice today to add about 0.1 per cent. Al to the cold mold, or when half filled, to quiet the steel. Carborundum is occasionally used to furnish C and Si; wolframite and chrome ore are used for tungsten and chrome steels. Probably 60–75 per cent. of the C of the charcoal is taken up by the steel; it absorbs 0.25 per cent. C from a new, and 0.15 per cent. C from an old graphite crucible. With the addition of ferromanganese these figures are raised to 2.0 per cent. and 0.5 per cent.

The Furnaces. The crucible furnace, heated with anthracite or coke,[2] and the Nobel furnace,[3] heated with petroleum, are found occasionally; the furnace most commonly used is that of Siemens, heated with producer or natural gas and shown in Figure 277.[4] The general arrangement of the regenerative chambers and of the reversing valves is the same as in the open-hearth furnaces. The oblong hearth is divided by vertical cross-partitions (which also support the roof) into a number of compartments. A "battery" thus has six to ten "holes," 3 x 2 feet and 5 feet deep, each of which holds six crucibles (pots), which, as shown in the drawing, are placed in two rows. Each hole is covered with a lid raised by a lever. In the bottom of a hole is an opening (8 inches in diameter) to run off any slag; the bottom is covered with a layer of coke (12 inches thick) on which rest the crucibles.[5]

The Crucibles[6] are of two kinds, clay and graphite. The latter are commonly used in this country. Clay crucibles are made of a finely ground mixture of raw fire clay with about 15 per cent. coke. A graphite crucible (Figure 278) is made of Ceylon graphite and air-dried fire clay. The proportions vary, 25–50 per cent. clay being used. The ingredients are mixed, ground in a roller mill, and the mixture allowed to come to a set, which takes a few days. The crucible is molded by pressing the mixture into a wooden mold, air dried and kiln burnt, when it loses

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[1] Ledebur, "Stahl und Eisen," 1885, p. 370.

[2] Percy, "Iron and Steel," p. 831; Harbord, "Steel," p. 228.

[3] Howe, "Steel," p. 302; "Revue Universelle des Mines," 1888, III, plate 8; "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1889, XXXVII, p. 81.

[4] Jeans, "Steel: Its History, Manufacture, Properties, and Uses," London, 1880, Spon, p. 337.

[5] Furnace of the Crescent Steel Company, Pittsburgh, Pennsylvania: "Iron Trade Review," May 19, 1898.

[6] Manufacture: Howe, "Steel," p. 298; Harbord, "Steel," p. 229; Ledebur, "Manual," 4th ed., p. 943; Walker, "Engineering and Mining Journal," 1895, LIX, p. 127; Stoermer, "Journal American Foundrymen's Association," 1901–1902, X[8], p. 53; Longmuir, "Iron Trade Review," April 3, 1902.

10 per cent. in weight. A crucible with a capacity of 100 pounds is $14\frac{1}{2}$ inches high, has an outside diameter of $10\frac{3}{4}$ inches at the top, $11\frac{9}{16}$ inches at the belly, $8\frac{1}{8}$ inches at the bottom; it is $1\frac{1}{2}$ inches thick at the bottom, $1\frac{1}{4}$ inches at the sides near the bottom, and $1\frac{1}{8}$ inches at the top; it weighs 28–30 pounds, is covered with a lid (cap) having a hole in the centre. It costs $1.25 and lasts four to six heats.

The Mold commonly used is a split mold (Figure 279),[1] held together by two rings and keys. It is smoked before using. An improved form is that of Sweet.[2]

The Tongs are of two kinds, Pull-out Tongs (Figure 280) and Teeming Tongs (Figure 281).

Crucible Steel Plants. General: Seebohm, "Journal Iron and Steel Institute," 1884, II, p. 372; Boeker, "Stahl und Eisen," 1886, p. 33; Thallner, "Zeitschrift des Vereins deutscher Ingenieure," 1900, p. 422. Special: Colonial Steel Company, "Iron Trade Review," May 7, 1903; Braun Söhne, "Oesterreichische Zeitschrift für Berg- und Huttenwesen," Harpf, 1899, p. 253. Styria and Austria: Ledebur, "Stahl und Eisen," 1895, p. 1; "Berg- und Hüttenmännische Zeitung," 1895, p. 103. Russia: Sergius, "Iron," 1892, XXXIX, pp. 271, 316, 335, 554. For steel projectiles: Sergius, "Journal Iron and Steel Institute," 1895, I, p. 194.

The Operation and Chemistry.[3] In American practice the crucible is charged cold, the larger pieces of bar iron are placed on the bottom and the rest of the charge poured on top. Every charge is made a little smaller than the preceding one, as the slag eating into the crucibles weakens it at the surface of the metal. A melt lasts three to four hours. When a charge is about melted, the cover is removed and the crucible content examined with an iron rod to see if everything is melted. When melted, the charge boils from the CO that is given off. The temperature is kept so that the boil shall not become violent. The next step is that of "killing" the metal, *i. e.*, holding it molten in the crucible until it becomes quiet.[4] This lasts about forty-five minutes, the quieting being caused by the metal taking up Si from the crucible ($SiO_2 + 2C = Si + 2CO$), which decreases its power to dissolve gas. Killed steel contains 0.2–0.4 per cent. Si. If the killing is carried too far, the metal becomes "dead," very quiet, and yields solid ingots; but they are brittle (having taken up too much Si). At the proper time and at the proper temperature, to be determined by the judgment of the melter, the metal is "teemed" (poured). One man removes the slag, a second one pulls out the crucible and places in the teeming tongs handled by two other men. In pouring, spattering must be avoided.

The Ingot, which is piped at the upper end, is "topped," *i. e.*, the piped part (20–30 per cent. of the weight of the ingot) is nicked and broken off and remelted. The length of the pipe is much reduced at some works (*e. g.*, Syracuse, New York) by not filling the mold in one pour, but stopping a moment when nearly filled, placing a fire-clay sleeve over the mold, and filling the mold and sleeve with steel. The steel in the sleeve cools more slowly than that in the mold and fills most of the space ordinarily occupied by the pipe. The ingot is "graded" by the fracture. When the C is between 1.0 and 1.5 per cent., differences of 0.05 per cent. can be recognized by the eye. The ingots are marked and stacked, ready to be hammered into flats, squares, octagonal bars, etc. The slag formed contains SiO_2, 40–45; FeO, 1–5; MnO, 18–30; Al_2O_3, 18–36.

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[1] Howe, "Steel," p. 305.

[2] "Iron Trade Review," July 7, 1904.

[3] Müller, "Stahl und Eisen," 1883, p. 603; 1885, p. 179; 1886, p. 695.

[4] Müller, *op. cit.*, 1885, p. 182; 1886, p. 698.

The loss in metal is very small, 2 per cent. and less. For a six-hole battery, five melters and four helpers are required.

Crucible Steel, being made from high grade raw material in a closed vessel where it is protected from S, is superior to Bessemer and open-hearth steel, as it is low in P and S, free from oxide, and contains hardly any gas. It runs high in C, 0.50–1.50 per cent.; low in P, <0.02; S, <0.04; Mn, <0.20; Si is high, <0.50; Cu, <0.20. Crucible steel furnishes the finest grades of cutlery and machine tools. According to the percentage of C, crucible steel is classed under fifteen grades or tempers, each of which has its specific uses. The following are some of the leading ones.[1]

C, per cent.	Use of steel for:
0.50–0.60	Hot work, battering tools.
0.60–0.70	Hot work, battering tools, tools of dull edge.
0.70–0.80	Battering tools, cold sets, reamers, taps.
0.80–0.90	Cold sets, hand chisels, drills, taps, reamers, dies.
0.90–1.00	Chisels, drills, dies, axes, knives, and similar purposes.
1.00–1.10	Axes, hatchets, large lathe tools, many kinds of dies and drills, if care be used in tempering them.
1.10–1.50	Lathe tools, graving tools, scribers, scrapers, little drills, and similar purposes.
0.90–1.10	*Best all-round tool steel.*

SECTION 83. Analyses of Crucible Steels.[2]

J. W. CABOT.

Grade.	C	P	Si	Mn	S
Ax	0.95	0.02	0.08	0.38	. . .
Rock drill	1.00	. . .	. . .	0.29	. . .
Files	1.40	0.03	0.25	. . .	0.01
Guns — Krupp	0.50	0.04	0.11	0.16	0.03
Projectiles — Whitworth	0.80	. . .	0.06	0.43	. . .
Projectiles — Krupp	0.55	0.03	0.23	0.32	0.008
Projectiles — Hotchkiss	0.55	0.05	0.05	. . .	0.05
Jessup's saws	1.00	0.025	0.19	0.30	. . .
Tool steel — Swedish	0.70	0.025	0.03	0.26	. . .
Tool steel — Midvale, U. S.	1.00	0.027	0.10	0.27	0.015
Tool steel — German	0.92	0.02	0.09	0.12	. . .
Pure iron — Swedish (specific gravity, 7.863)	0.07	0.02	0.04	0.03	0.02

[1] Metcalf, "Steel: A Manual for Steel Users." Wiley, New York, 1896, p. 21.

[2] Richards, "Notes on Iron," p. 127.

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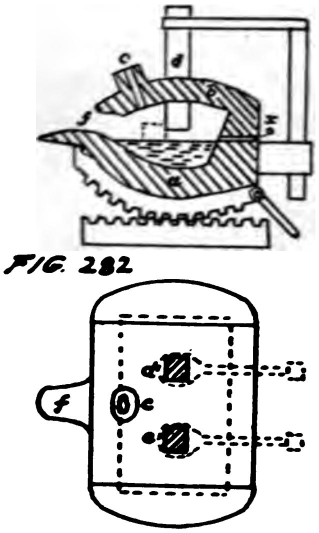
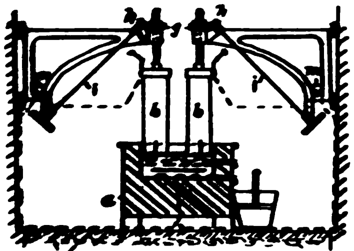


FIG. 282

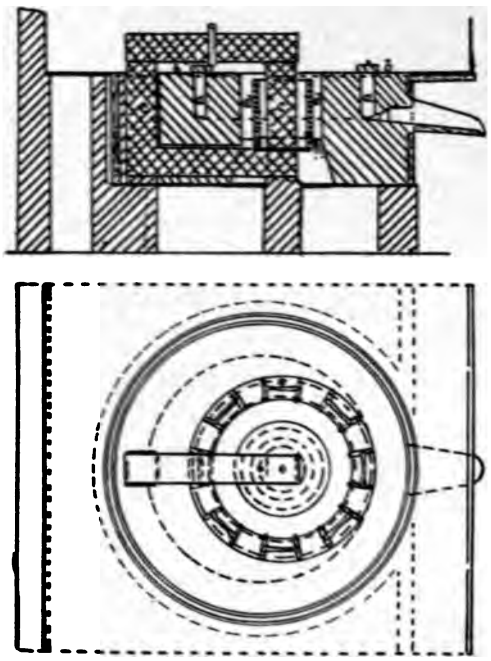
HÉROULT OSCILLATING STEEL FURNACE

FIG 283



HÉROULT STATIONARY STEEL FURNACE

FIG 285



KJELLIN STEEL FURNACE

FIG 293

A list of articles made of crucible steel, arranged according to their C-content, is published in "Iron Age," July 10, 1902, p. 8.

H. ELECTRIC PRODUCTION OF STEEL.

Literature: Borchers, "Electrometallurgie," 3d ed., p. 530; Garnier, "Electrosidérurgie," Paris, 1903.

SECTION 84. **General.** In Section 35 the production of pig iron from iron ore by electric smelting was spoken of as being hopeless unless a. special brand of pig iron was to be produced which was to be converted into a very high grade steel. The electric conversion of pig iron into steel, based upon open-hearth principles, or the electric production of steel along lines similar to those of making crucible steel, are more promising. In fact, processes of this class are in successful operation. The first experiment of making steel electrically was that by Sir W. Siemens in 1878,[1] who embedded a graphite crucible (with positive carbon pole passing through its bottom) in a sheet-iron box filled with powdered charcoal or other refractory material, inserted an adjustable negative carbon pole through a hole in the lid and started an electric current. This, entering by the positive pole, passed through the material to be fused, and, jumping the air space between this and the negative pole, formed an arc and fused the charge. Later he replaced the positive carbon pole by a water-cooled copper tube. Steel made by this method was unsatisfactory, as it was found difficult, if not impossible, to control the carbon content and to exclude oxidation. In the beginning of the nineties the matter was revived by Laval, Taussig, Urbanitzky-Fellner, and others, but failed again on account of the unsatisfactory character of the product.

SECTION 85. **Modern Furnaces.** Of the modern furnaces three may be discussed, those of Héroult, Gin, and Kjellin.

(1) *The Héroult Furnaces.*[2] These furnaces belong to the resistance type. The Oscillating Furnace (Figure 282) is similar in general outline to the Wellman tilting open-hearth furnace (page 173). It consists of a tilting boiler-iron casing *a* lined with refractory material free from carbon, a lined cover *b* with chimney *e*, a pouring spout *f*, a tuyère *H* to permit Bessemerizing, and two carbon electrodes *d* that can be raised and lowered at will. In the sketch of the Stationary Furnace (Figure 283) are shown the supports of the electrodes by cantilever frames *k*, the manner of raising and lowering them by hand wheels *x* and gearing *g* and *h*. The furnace *a* is without a roof; it has a metal and a slag tap; the metal is covered by a layer of slag; the current travels in the direction of the arrow. In starting, the furnace is filled with cold or melted mixture of pig iron and scrap steel, covered with slag, and then the current is turned on which heats up the charge and brings it to complete fusion. The slag floating on the metal prevents any contact between carbon electrode and metal. In order to force the current to travel in the direction of the arrow, it is essential that the electrodes be apart the correct distance, and that they dip sufficiently into the slag to bring the slag between the ends of the electrodes and metal bath to such a heat that its resistance to the passage of the current shall be less than that of the cooler surface of the slag between the sides of the electrodes. It

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[1] "Electro-technische Zeitschrift," 1880, I, p. 325.

[2] "Electro-chemical Industry," I, pp. 63, 287, 459; "Electrical World and Engineer," September 6, 1902; "Engineering News," May 2, 1901; "Jahrbuch der Electro-chemie," 1901, p. 552.

is easily seen that, having a metal bath maintained at a desired temperature and protected by a fused slag, oxidizing or reducing agents can be used to change the character of the metal, and metals or alloys introduced to form special steels. At La Praz, Héroult produces from cold pig iron and scrap 44 pounds steel per electric horse power per day.[1]

(2) *The Gin Furnace.*[2] This resistance furnace represented in horizontal section in Figure 284 has so far been used only in an experimental way, but the results have been such as to promise well for the production of steel. It consists of a hearth, *A*, of refractory material with a serpentine trough of small cross section. At the ends of the trough are large water-cooled blocks of steel, *B*, forming the terminals of the circuit, and at the bends are tap holes to remove the steel. Steel is made by the pig and ore, or the pig, scrap, and ore process. The current passing from the large-section water-cooled terminals into the sinuous trough of small section melts the charge. The Si, Mn, C, etc., are oxidized by the action of the solid O of the ore and not by atmospheric O; recarburization is effected in the usual way. It is claimed that with power at $20 per horse power year steel can be made more cheaply than by the open-hearth method.

(3) *The Kjellin Furnace.*[3] The furnace (Figure 285) is an induction furnace. It consists of an annular groove or hearth, *AA*, of refractory material and a roof, *BB*. A quadrangular core *C*, made up of thin insulated sheets of soft iron, encloses a small part of the hearth; a coil of insulated copper wire, *DD*, connected with an alternating current dynamo encircles one limb of the core. The current passing through the coil *DD* excites a varying magnetic flux in the core *CC*, and this induces an alternating electric current in the contents of hearth *AA* and melts the charge. The arrangement is similar to a step-down transformer having a large number of primary turns (in *DD*) and a single secondary turn (in *AA*).

In transformers

$$\frac{\text{E. M. F. in secondary current}}{\text{E. M. F. in primary current}} = \frac{\text{turns in secondary coil.}}{\text{turns in primary coil.}}$$

The number of turns in a primary coil is governed by the voltage of the alternating dynamo. The current that may be drawn from the secondary coil is in inverse proportion to the voltage of the secondary: —

$$\text{Primary } C : \text{secondary C} = \text{secondary } E : \text{primary } E, \text{ or}$$
$$\text{Primary } C \times \text{primary } E = \text{secondary } C \times \text{secondary } E.$$

If the primary coil takes 50 amperes at 3,000 volts and has 600 turns, the secondary coil with one turn will have an E. M. F. of 5 volts, and the current in it will be of 30,000 amperes; the efficiency 96 + per cent. By this arrangement, an alternating circuit of high tension is changed into a heavy current of low voltage without the use of a transformer or of copper cables of large section; the expensive and uneconomical carbon electrodes are avoided, and the metal does not come into contact with anything that may introduce any impurities.

The furnace at Gysinge, Sweden, holds 1,800 kilograms steel; 1,000–1,100 kilograms are removed with each tap; the rest is left behind to keep the current passing. It produces in

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[1] Borchers, "Electrometallurgie," 1903, p. 543.

[2] "Electro-chemist and Metallurgist," 1904, III, p. 572; Bennie, "Electro-chemical Industry," 1904, p. 20; Richards, "Technology Quarterly," 1904, XVII, p. 28; "Berg- und Hüttenmannische Zeitung," 1904, p. 52.

[3] "Transactions American Institute of Mining Engineers," 1903; "Electro-chemical Industry," 1903, I, p. 576.

twenty-four hours by the pig and scrap process, using a cold charge, 4,100 kilograms steel, with an expenditure of 165 kilowatts or 225 electric horse power. The generator gives an alternating single-phase current of 3,000 volts, which is transformed into a current of about 30,000 amperes. The losses have been proved experimentally to be 87.5 kilowatts, so that the effective power absorbed by the steel is 165 − 87.5 = 77.5 kilowatts; thus 1 effective kilowatt day produces about 53 kilograms = 116.6 pounds steel. It has been calculated that a furnace of 736 kilowatts, or 1,000 electric horse power, will produce 30,000 kilograms steel ingots in twenty-four hours when charged with cold materials.

I. INFLUENCE OF CERTAIN ELEMENTS UPON THE MECHANICAL PROPERTIES OF STEEL. SPECIAL OR ALLOY STEELS.

Literature: Howe, "Metallurgy of Steel," p. 13 and following; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 456 and following; von Jüptner, "Siderologie," Leipsic, 1902, II, p. 198.

SECTION 86. **Influence of Carbon on Unhardened Steel.**[1] The following tables are taken from Howe's "Metallurgy of Steel," pp. 16 and 17. The data relating to tensile strength have been represented graphically in Figures 24 and 131.

(1) *Tensile Strength.*

C, per cent.	0.05	0.10	0.15	0.20	0.30	0.40	0.50	0.60	0.80	1.00	1.30
Upper limit, lbs. per square in. . . .	66,000	70,000	75,000	80,000	90,000	100,000	110,000	120,000	150,000	170,000	115,000
Lower limit, lbs. per square in. . . .	50,000	50,000	55,000	60,000	65,000	70,000	75,000	80,000	90,000	90,000	90,000

The tensile strength of unhardened steel increases up to about 1 per cent. C and then decreases.

(2) *Elongation.*

C, per cent.	0.10	0.20	0.30	0.40	0.50	0.60	0.70	0.80
Usual upper limit, per cent. . . .	29.0	25.2	23.0	21.0	. . .	10.0	7.5	6.0
Usual lower limit, per cent. . . .	17.5	15.0	12.0	. . .	7.5	. . .	2.5	1.5

The elongation is at its maximum with carbonless iron; it decreases with an increase of C, slowly with C, 0.0–0.2 per cent., more quickly with 0.2–0.5 per cent., and then again more slowly.

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[1] C in general, see p. 4.

(3) *Tensile Strength for Given Elongation.*

Elongation, per cent.		4	6	8	10	12	14	16
Usual limits of tensile strength.	Upper limit, lbs. per square in.	150,000	138,000	130,000	122,000	115,000	108,000	102,000
	Lower limit, lbs. per square in.	111,000	102,000	94,000	85,000	80,000	75,000	70,000

Elongation, per cent.		18	20	22	24	26	28	30	32
Usual limits of tensile strength.	Upper limit, lbs. per square in.	98,000	93,000	88,000	80,000	77,000	74,000	71,000	70,000
	Lower limit, lbs. per square in.	64,000	59,000	56,000	52,000	51,000	51,000	50,000	50,000

The tensile strength decreases as the elongation increases.

(4) *Compressive Strength.*

C, per cent.	0.3	0.6	0.9	1.2	Remarks.
Ultimate compressive strength, lbs. per square in.	47,513	84,827	95,207	102,173	The figures refer to pieces of a length = 8 × the diameters and probably square.
Elastic compressive strength, lbs. per square in.	40,666	52,666	58,000	61,666	

The compressive strength rises with increasing C within limits as yet unknown.

SECTION 87. **Influence of Si on Unhardened Steel.**[1]

(1) *Tensile Strength and Elongation.* The following table is part of a more extended one given by Howe.[2]

C	Si = 0.0–0.05		Si = 0.20–0.30	
	Tensile strength.	Elongation.	Tensile strength.	Elongation.
	Lbs. per square inch.	Per cent.	Lbs. per square inch.	Per cent.
0.00–0.10	59,181	27.0	55,000	25.0
0.10–0.20	60,636	26.0	74,000	24.4
0.20–0.30	70,000	23.0	67,000	21.0
0.40–0.50	81,600	8.6	104,300	12.3
0.50–0.60	88,500	3.0	130,000	14.0
0.70–0.80	126,000	4.4	131,300	3.4
0.80–0.90	149,000	3.0	139,000	8.0
0.90–1.00	134,000	2.0	130,000	5.0

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[1] Si in general, see p. 14; Harbord, "Steel," p. 597; Turner, "Journal Chemical Society," 1887, p. 129; Hadfield, "Journal Iron and Steel Institute," 1889, II, p. 222.

[2] "Steel," p. 39.

The tensile strength rises until C has reached 0.9 per cent., and then declines; the elongation falls almost continuously.

(2) *Tensile Strength of Silicon Steels.*[1]

COMPOSITION.					Ultimate strength.	Elastic limit.	Elastic ratio	Elongation in 2 inches.	Reduction of area.	Ultimate strength after annealing.
C	Si	Mn	S	P						
Per cent.	Per cent.	Per cent.	Per cent.	Per cent.	Pounds per square inch.	Pounds per square inch.	Per cent.	Per cent.	Per cent.	Pounds per square inch.
0.14	0.21	0.14	0.08	0.05	73,920	49,280	66.7	30.05	54.54	56,000
0.18	0.77	0.21	...	...	76,160	56,000	73.5	29.50	54.54	64,960
0.19	1.57	0.28	...	...	84,000	62,720	74.7	31.10	50.58	73,920
0.20	2.14	0.25	0.06	0.04	88,480	69,440	78.5	18.48	28.02	76,160
0.20	2.67	0.25	...	...	95,200	71,680	75.3	17.60	24.36	71,680
0.21	3.40	0.29	...	...	106,400	78,400	73.7	11.10	14.22	87,360
0.25	4.30	0.36	...	...	109,760	100,800	91.8	0.004	0.20	85,120
0.26	5.08	0.29	0.06	0.04	107,520	Not visible	...	0.30	0.70	56,000

The data show that Si has no such decided influence upon the physical properties of steel as have P, S, or C. This is shown in a more compact form by the following two extreme examples:[2] —

COMPOSITION.					Ultimate strength.	Elastic limit.	Elastic ratio.	Elongation in 10 inches.	Reduction of area.
C	Si	Mn	S	P					
					Lbs. per square in.	Lbs. per square in.	Per cent.	Per cent.	Per cent.
0.170	0.056	0.576	0.061	0.058	69,675	49,106	70.5	21.5	46.3
0.160	0.402	0.493	0.059	0.096	78,863	54,798	69.5	18.0	33.9

See also Guillet, "Revue de Métallurgie," 1904, p. 47, or "Les Aciers Spéciaux," Dunod, Paris, 1904.

SECTION 88. **Influence of Manganese.**[3]

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[1] Hadfield, "Journal Iron and Steel Institute," 1889, II, p. 222; through Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 457.

[2] Campbell, *op. cit.*, p. 460.

[3] Manganese in general, see p. 14; Hadfield, "Journal Iron and Steel Institute," 1888, II, p. 70; Souther, "Mineral Industry," IX, p. 419; Guillet, "Bulletin Société d'Encouragement," November, 1903; (Ledebur, "Stahl und Eisen," 1904, p. 281); "Revue de Métallurgie," 1904, p. 281; "Les Aciers Spéciaux," Dunod, Paris, 1904.

COMPARATIVE PHYSICAL PROPERTIES OF OPEN-HEARTH STEELS WITH DIFFERENT CONTENTS OF MANGANESE.[1]

Kind of steel.	Limits of ultimate strength.	P	Mn	Ultimate strength.	Elastic limit.	Elongation in 8 inches.	Reduction of area.	Elastic ratio.	Shape of test piece in inches.
	Pounds per square inch.	Per cent.	Per cent.	Pounds per square inch.	Pounds per square inch.	Per cent.	Per cent.	Per cent.	
Acid	55,000 to	0.08	0.30	57,922	38,692	29.91	59.02	66.8	¾ diameter
	60,000		0.37	58,881	38,598	28.08	57.07	65.6	
Basic	55,000 to	0.03	0.44	58,005	38,547	30.16	60.21	66.5	2 x ⅜
	63,000		0.57	59,563	40,133	30.36	58.55	67.4	
Acid	60,000 to	0.08	0.35	62,180	41,308	28.00	50.89	66.4	¾ diameter
	65,000		0.51	62,605	41,169	27.65	54.66	65.8	
Acid	65,000 to	0.08	0.51	67,421	43,923	25.96	51.29	65.1	¾ diameter
	70,000		0.78	68,192	45,854	25.82	51.50	67.2	
Acid	70,000 to	0.08	0.60	72,353	46,836	24.23	47.79	64.7	¾ diameter
	75,000		0.91	72,115	48,359	24.63	47.73	67.1	
Acid	75,000 to	0.08	0.65	77,520	49,411	22.34	44.42	63.7	¾ diameter
	80,000		0.84	78,083	50,226	23.63	48.49	64.3	
Acid	80,000 to	0.08	0.68	81,747	51,219	20.63	41.04	62.7	¾ diameter
	85,000		0.82	81,860	52,231	22.67	47.75	63.8	
Acid	85,000 to	0.08	0.75	86,460	54,517	20.41	40.56	63.1	¾ diameter
	90,000		0.83	88,034	55,409	20.66	41.92	62.9	

Manganese has little effect on the tensile strength and ductility of steel, but it does affect its resistance to shock, and for this reason it is advisable to keep it as low as possible without interfering with good working in the rolls.

Manganese Steels.[2] As Mn rises above 1.5–2.0 per cent. the steel becomes brittle and worthless; when it reaches 8 per cent. it suddenly becomes ductile; between 8 and 10 per cent. it is extremely hard, tough, practically unmachinable, and non-magnetic; it is made into cutting tools which require no quenching; quenching does not harden it, annealing does not soften it.

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[1] Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 465.

[2] Howe, "Steel," pp. 48, 361.

PHYSICAL PROPERTIES OF FORGED MANGANESE STEEL WITH 0.83–21.69 PER CENT. Mn.[1]

No. of sample.	COMPOSITION.			NATURAL.		QUENCHED IN WATER.		ANNEALED.	
	C	Si	Mn	Ultimate strength.	Elongation in 8 inches.	Ultimate strength.	Elongation in 8 inches.	Ultimate strength.	Elongation in 8 inches.
	Per cent.	Per cent.	Per cent.	Pounds per square inch.	Per cent.	Pounds per square inch.	Per cent.	Pounds per square inch.	Per cent.
1	0.20	0.03	0.83	73,920	31	. . .	. . .	. . .	. . .
2	0.40	0.15	2.30	125,440	6	. . .	. . .	. . .	. . .
3	0.40	0.09	3.89	85,120	1	. . .	. . .	. . .	. . .
4	0.52	0.37	6.95	56,000	2	51,520	2	47,040	2
5	0.47	0.44	7.22	60,480	2	56,000	2	60,480	5
6	0.61	0.30	9.37	73,920	5	87,360	15	85,120	16
7	0.85	0.28	10.60	76,160	4	89,600	17	91,840	17
8	1.10	0.16	12.60	87,360	2	120,960	27	82,880	11
9	0.92	0.42	12.81	87,360	5	136,640	37	107,520	20
10	0.85	0.28	14.01	80,640	2	150,080	44	107,520	14
11	1.10	0.32	14.48	87,360	1	141,120	37	109,760	5
12	1.24	0.16	15.06	109,760	2	136,640	31	105,280	2
13	1.54	0.16	18.40	114,240	1	118,720	10	87,360	1
14	1.83	0.26	18.55	96,320	1	123,200	5	. . .	. . .
15	1.60	0.26	19.10	116,480	1	132,160	4	91,840	1

The great disadvantage of Mn steel is that it is so hard that it cannot be machined even with the best of hardened tools. It is readily forged between dull and light redness; cold forging makes it brittle; its toughness is restored by heating.

SECTION 89. **Influence of S.**[2] S injures the rolling qualities of steel, especially at a dull red heat, causing it to crack and tear, and lessening its capacity to weld. These bad effects can be overcome to some extent by the addition of Mn,[3] and by care in heating. In ordinary practice S varies from 0.02 to 0.10 per cent. With common steel and simple shapes it can reach 0.10 per cent. With rails and shapes having thin flanges it should be below 0.08 per cent.

SECTION 90. **Influence of Cu.**[4] The main effect of Cu upon steel is to make it red-short and to destroy the welding power, but steel may contain as much as 1 per cent. Cu without

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[1] Hadfield, "Journal Iron and Steel Institute," 1888, II, p. 70; through Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 468.

[2] S in general, see p. 16; Howe, "Steel," p. 48; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 468.

[3] Arnold-Waterhouse, S and Mn in Steel: "Journal Iron and Steel Institute," 1902, I, p. 136.

[4] Cu in general, see p. 16; Howe, "Steel," p. 82; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 472; Ball-Wingham, "Journal Iron and Steel Institute," 1889, I, p. 223; Campbell, "Journal Iron and Steel Institute," 1897, II, p. 80; 1903, II, p. 338; Stead, *op. cit.*, 1901, II, pp. 89, 104, 122; Colby, "Iron Age," November 30, 1899.

having any harmful effect, provided S is not high. If, *e.g.*, 0.08–0.10 per cent. S be present at the same time, the effect of the two will be cumulative, and the steel will crack when it is being rolled. In the cold the physical properties are the same as if Cu were absent. Steel with up to 0.5 per cent. Cu will weld unsatisfactorily when associated with ordinary amount of S, such as 0.06–0.08 per cent.

SECTION 91. **Influence of P.**[1] P inclines to produce a coarse crystallization of steel which diminishes malleableness at high temperatures; hence phosphoritic steel has to be rolled (finished) at a lower temperature than pure steel. C intensifies the bad effects of P; Si does the same, only in a lower degree. P increases the static strength up to 0.12 per cent.; it greatly reduces the resistance to shock, increases the elastic limit, and reduces the elongation and contraction. While the ordinary testing machine shows only small differences within the common P content ranging from 0.04 to 0.10 per cent., a high-P steel under shock will prove to be brittle, while one low in P will be tough.

SECTION 92. **Influence of Al.**[2] While Al is used to a considerable extent to quiet steel, little combines with the steel itself. It has little effect upon the tensile strength; the ductility is not diminished until the Al content rises to 2 per cent.

SECTION 93. **Influence of Ni.**[3] Ni alloys with iron in all amounts up to 50 per cent. Alloys with less that 8 per cent. Ni have a greater tensile strength than the corresponding C steels and the same toughness. Alloys with 8–22 per cent. Ni are brittle. Alloys with over 22 per cent. Ni are non-magnetic, non-corrosive, possess a low coefficient of expansion — properties which make them suited for scientific instruments. Ordinary nickel steel is a low-carbon (under 0.4 per cent.) steel with about 3.25 per cent. Ni, 0.3–0.6 per cent. Mn, 0.02 per cent. P It has a high density, elastic limit, ultimate strength and ductility, is readily forged; its welding power decreases with the increase of Ni; it is practically non-corrodible. On account of the cost of Ni, its use is restricted, *e.g.*, to bicycle tubing and spokes, motor carriages, solid and hollow shafting, reciprocating parts of machinery, armor plate, guns, etc. The New York Central and Hudson River Railroad[4] is experimenting with Ni steel rails (C, 0.418; Si, 1.02; Mn, 0.79; Ni, 3.38; P, 0.084) on part of its line where, with a steep grade, there is a heavy traffic. Ni steel has been used for the tension members of the new cantilever bridge, Blackwell's Island, New York City.

The following table of Campbell,[5] reproduced only in part, brings out the leading differences between Ni steels with 3.25 per cent. Ni and C steels of similar general chemical composition made in the acid open-hearth furnace: —

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[1] P in general, see p. 15; Howe, "Steel," p. 67; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 469.

[2] Harbord, "Steel," p. 616; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 475; Hadfield, "Journal Iron and Steel Institute," 1890, II, p. 161.

[3] Howe, "Steel," pp. 86, 369; Harbord, "Steel," p. 623; Campbell, "Manufacture and Properties of Iron and Steel," 1903, p. 479; Riley, "Journal Iron and Steel Institute," 1889, I, p. 45; Browne, "Transactions American Institute of Mining Engineers," XXIX, p. 569; Rudeloff, "Zeitschrift des Vereins zur Beförderung des Gewerbefleisses," 1898, p. 348; 1902, p. 348; Vogel, "Stahl und Eisen," 1895, p. 718; Wedding, "Stahl und Eisen," 1902, p. 1287; Colby, "Iron Trade Review," November 14, 1901; "Iron Age," August 20, 1903; "Proceedings American Society for Testing Materials," III, p. 141; Porter, "Cassier's Magazine," August, 1902; Campbell, "American Society of Civil Engineers," June, 1895: Souther, "Mineral Industry," 1900, IX, p. 421; Dumas, "Bulletin de l'Industrie Minérale," 1903, II, p. 641; Osmond-Cartaud, "Revue de Métallurgie," 1904, p. 69; Boudouard, *op. cit.*, p. 80; Guillet, "Les Aciers Spéciaux," Dunod, Paris, 1904.

[4] "Iron and Steel Magazine," I, p. 201.

[5] "Manufacture and Properties of Iron and Steel," 1903, p. 479.

Kind of steel.	COMPOSITION, PER CENT.				
	C	Mn	P	S	Ni
Nickel	0.24	0.78	0.032	0.027	3.25
Hard forging	0.30–0.35	0.60–1.00	0.03–0.05	0.03–0.05	None
Forging	0.25–0.30	0.60–0.80	0.03–0.06	0.03–0.07	None

Shape of Member.	Kind of steel.	Ultimate strength.	Elastic limit.	Elastic ratio.	Elongation in 8 inches.	Elongation in 2 inches.	Reduction of area.
		Pounds per square inch.	Pounds per square inch.	Per cent.	Per cent.	Per cent.	Per cent.
Rounds	Nickel	86,015	63,575	73.9	20.19	34.00	46.3
	Hard forging	87,663	58,055	66.2	16.70	24.44	30.3
	Forging	78,066	51,793	66.3	23.94	. . .	52.0
Angles	Nickel	86,960	58,553	67.3	21.75	39.67	50.5
	Hard forging	87,820	54,153	61.7	19.25	34.83	43.3
	Forging	76,970	49,544	64.4	. . .	. . .	49.6

SECTION 94. **Influence of Cr.**[1] Chrome steel contains 0.5–3.0 per cent. Cr with 0.8–1.0 per cent. C. The combined presence of Cr and C makes the steel very hard; its forgableness is similar to carbon steel of like hardness. It is very fine grained. Chrome has a strong tendency to oxidize. On account of its hardness and its high resistance to shock, chrome steel is used for burglar-proof safes, armor plate, projectiles, etc. Steel with about 1 per cent. C and 1.5–2 per cent. Cr gives a tool of extraordinary hardness. Cr is usually associated with W in self-hardening steels.

ANALYSES OF CHROME STEELS.[2]

	C	Cr	Mn	Si
Unieux	0.60	2.20	. . .	. . .
Brooklyn	0.63	1.04	0.05	0.15
Brooklyn	0.44	0.92	0.03	0.12
Brooklyn	0.29	1.32	0.15	0.15

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[1] Moissan, "Le Four Électrique," Paris, 1897, p. 206; Le Blanc, "Die Darstellung des Chroms und seiner Verbindungen," Halle, 1902; Howe, "Steel," pp. 75, 366; Harbord, "Steel," p. 618; Hadfield, "Journal Iron and Steel Institute," 1892, II, p. 49; Benneville, *op. cit.*, 1896, I, p. 222; Guillet, "Revue de Métallurgie," 1904, p. 155.

[2] Ledebur, "Manual," 4th ed., p. 727.

Ferrochrome runs high in C; thus a 65 per cent. alloy[1] contained 12 per cent. C, probably all in the combined state; 80 per cent. ferrochrome is an article of commerce. In the last few years a refined ferrochrome with 65–70 per cent. Cr and only 0.5–1.0 per cent. C has come into the market. This has made it possible to make high-Cr and low-C steels. The following table[2] gives the mechanical properties of steels with varying percentages of Cr, but with constant C of about 0.4 per cent.: —

Cr Per cent.	Condition.	Elastic limit, tons per square inch.	Breaking stress, tons per square inch.	Elongation, per cent.	Contraction, ratio of fractional to original surface.
5	Annealed	17.8	31.8	24.0	0.240
5	Hardened and tempered	48.8	55.2	12.0	0.370
10	Annealed	22.9	42.0	21.5	0.440
10	Hardened and tempered	42.4	54.3	12.0	0.536
15	Annealed	25.4	45.3	18.5	0.500
15	Hardened and tempered	48.8	58.1	11.5	0.546
20	Annealed	21.2	36.1	21.5	0.465
20	Hardened and tempered	27.6	40.3	19.5	0.515
25	Annealed	29.8	42.2	18.0	0.621
25	Hardened	27.6	40.4	20.0	0.500
30	Annealed	31.8	41.6	19.0	0.620
30	Hardened	28.8	39.0	19.0	0.650

Ni-Cr steels[3] are used in the manufacture of armor plate.

SECTION 95. **Influence of W.**[4] W imparts hardness to steel, but in a less degree than Cr; W steels, therefore, receive often 0.25–3.50 per cent. Cr. Tungsten (Mushet) steel contains 2–11, usually 4–6, per cent. W, 2–3 per cent. Mn, 0.5–1.5 per cent. C. It is extremely hard (cannot be machined), very brittle, can be forged with difficulty only between a cherry red and a low yellow heat. It is liable to crack upon hardening; happily it does not need it, as it is self-hardening. It is used for tools having a dull edge, for car and locomotive springs, shoes of stamp mills, etc.

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[1] "Journal Iron and Steel Institute," 1892, II, p. 70.

[2] "Mineral Industry," IX, p. 419.

[3] Harbord, "Steel," p. 627; Vogel, "Stahl und Eisen," 1895, p. 721.

[4] Howe, "Steel," p. 81; Harbord, "Steel," p. 630; Wahl, "Engineering and Mining Journal," 1892, LIV, p. 610; Nau, "Iron Age," 1892, XLIX, p. 248; Benneville, "Journal Iron and Steel Institute," 1896, I, p. 222; Hadfield, *op. cit.*, 1903, II, p. 14; Metcalf, Alloy Steels: "American Society for Testing Materials," May, 1904; Guillet, "Revue de Métallurgie," 1904, p. 263.

ANALYSES OF SELF-HARDENING STEEL.[1]

C	Si	S	P	Mn	W
2.30	1.05	. . .	. . .	2.57	6.12
2.00	1.60	0.02	0.04	1.72	8.22
2.05	0.79	0.04	0.04	2.30	8.04
1.67	0.33	. . .	. . .	2.53	5.74
2.35	0.15	. . .	. . .	3.38	11.02

Taylor-White High-speed Cutting Steel.[2] This is a self-hardening, specially tempered alloy steel made by the Bethlehem Steel Company, South Bethlehem, Pennsylvania. It contains Cr-W or Cr-Mo alloys; the range of composition is from Cr, 0.75, W, 4.00, Mo, trace, to Cr, 3.00, W, 8.00, and varying Mo. On account of the great possibilities, many steel makers have experimented to produce something similar. There are about fifty different brands in the market. The composition of two leading brands outside of the Taylor-White is: W, 9.99; Cr, 2.83; C, 0.60; P, 0.010; S, 0.010; Si, trace; Mn, trace; and W, 18.48; Cr, 2.90; C, 0.79; P, S, Si, not determined; Mn, 0.33.[3] The enormous cutting speed of 150 feet per minute[4] is obtained with mild steel castings, the cutting edge becoming red-hot and the chips tarnished blue. In tempering, the steel is heated to 1,000° C., rapidly cooled in a bath of molten lead, kept there for ten minutes, and then slowly cooled in lime. For some purposes the steel is reheated to from 370 to 750° C. when quite cold and then air-cooled. In all tempering operations the tool is protected against oxidation by a special flux. Osmond[5] had found, in 1892, that W-steel remained unhardened when it was heated to 830° C. and quenched at 630° C., but that it became very hard when it was brought to a higher (1,310° C.) and quenched at a lower (555° C.) temperature. His results were made public only in 1903.

SECTION 96. **Influence of Mo.**[6] Mo resembles W in its effect upon steel in that it increases the elongation and the resistance to shock; its effect is twice as powerful as that of W, and Mo steels stand heat treatment better than W steels. Mo has been used experimentally in the manufacture of armor plate. It seems to be especially advantageous when used in connection with Ni, or Ni and Cr. Mo steels contain, usually, about 0.25 per cent. Mo. However, Metcalf (*l. c.*) publishes the following analysis: Mo, 9.65; Cr, 0.00; C, 0.66; P, 0.016; Si,

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[1] Hadfield, "Journal Iron and Steel Institute," 1903, II, p. 66.

[2] Day, "Journal Franklin Institute," 1901, CLII, p. 161; Report of Committee: *op. cit.*, 1903, CLV, p. 127; "Stahl und Eisen," 1903, p. 132; "Iron Age," September 19, 1901; December 31, 1903; Metcalf, *op. cit.*, June 23, 1904; Harbord, "Steel," p. 632: Hadfield, "Journal Iron and Steel Institute," 1903, II, p. 66; Metcalf, Alloy Steels: "American Society for Testing Materials," May, 1904.

[3] Metcalf, *l. c.*

[4] Kleinhans, Surface Speeds for Milling Cutters: "Iron Trade Review," August 6, 1903.

[5] "Journal Iron and Steel Institute," 1903, II, p. 65.

[6] Moissan, "Berg- und Hüttenmännische Zeitung," 1895, p. 301; Guichard, "Stahl und Eisen," 1896, p. 693; Lipin, *op. cit.*, 1897, p. 571; Helouis, "Berg- und Hüttenmännische Zeitung," 1897, p. 150; Helmhacker, "Engineering and Mining Journal," October 8, 1890; Souther, "Mineral Industry," 1900, IX, p. 421; Colby, "Iron Trade Review," November 14, 1901; Leo, "Berg- und Hüttenmännische Zeitung," 1904, p. 96, from "American Manufacturer and Iron World," 1903, p. 809.

0.046; Mn, 0.22. The alloy sold in the market contains Mo, 75–50 per cent.; Ni, 25–50 per cent.; Fe, 2.0–2.5 per cent.; C, 1.0–1.5 per cent.; Si, 0.25–0.50 per cent.

SECTION 97. **Influence of Va.**[1] An addition of $\frac{1}{3}$–$\frac{1}{2}$ per cent. of Va to steel more than doubles the elasticity of mild steel, and nearly doubles its tensile strength; it resembles W in its effect on tool steel, but is from ten to twenty times as powerful. Va steels acquire the maximum hardness when annealed at 700–800° C., which is of great importance when used in high-speed cutting. Guillet (*l. c.*) considers that the simple Va steels will be used as tool steels, and that the more complex alloys, such as Va–Ni, have more valuable properties than the simple ones. The high price of ferrovanadium ($12.50–$48.48 in 1901) per pound Va limits its use.

SECTION 98. **Influence of Ti.**[2] It is difficult to introduce Ti into steel, as it is very easily oxidized. Howe[3] could not find any Ti in an undoubted sample of Mushet's "titanic steel." Nevertheless, titanium steels with 1 per cent. and less Ti have been made in crucibles by the addition of ferrotitanium. It is said to increase the tensile strength without reducing the ductility, but little that is definite is known about it.

SECTION 99. **Calculation of the Influence of Certain Elements on Steel.** Investigators of the physical properties of iron and steel have constructed formulæ and figured out tables by means of which the effects of different percentages of the various elements can be foretold and thus steels compounded to satisfy given requirements. Chief amongst these are von Jüptner, "Beziehungen zwischen der Zerreissfähigkeit und Chemischen Zusammensetzung von Eisen und Stahl," Leipsic, 1895; "Beziehungen zwischen der Chemischen Zusammensetzung und den Physikalischen Eigenschaften von Eisen und Stahl," Leipsic, 1896; also "Siderologie", Leipsic, 1902, II, pp. 198–361; Webster, "Transactions American Institute of Mining Engineers," XXI, p. 766; XXIII, p. 113; XXVIII, p. 618; "Journal Iron and Steel Institute," 1894, I, p. 328; and Campbell, "Manufacture and Properties of Iron and Steel," New York, 1903, p. 482.

SECTION 100. **Influence of Gases.**[4] Liquid steel holds in solution gases in the same manner as water does O, CO_2, etc. The gases will pass off as the dissolving power decreases with the lowering of the temperature. Some of the gas will not be able to pass off entirely on account of the metal having become pasty; some, however, will remain occluded, being alloyed with the metal. The cases to be considered are N, CO, and H.

(1) *Nitrogen.* Basing the opinions on the early researches of Parry,[5] it was believed that steel retained very large quantities of N. Later researches, however, showed that this was not the case; Tholander[6] and Harbord-Twynam[7] found only 0.006–0.0022 per cent. N.

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[1] Harbord, "Steel," p. 636; Helouis, "Bulletin de la Société d'Encouragement," 1896, p. 904; "Berg- und Hüttenmännische Zeitung," 1897, pp. 149, 295; Vogel, "Stahl und Eisen," 1896, p. 615; "X, American Manufacturer and Iron World," 1897, p. 332; Editor, "Engineering and Mining Journal," August 18, 1900; Editor, "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1901, p. 98; Guillet, "Electro-chemist and Metallurgist," 1904, p. 609.

[2] Harbord, "Steel," p. 629; Rossi, "Mineral Industry," IX, p. 727; "Transactions American Institute of Mining Engineers," XXXIII, p. 179.

[3] "Steel," p. 85.

[4] Howe, "Steel"; Harbord, "Steel," p. 612; Ledebur, "Manual," 4th ed., p. 364; von Jüptner, "Siderologie," I, p. 234.

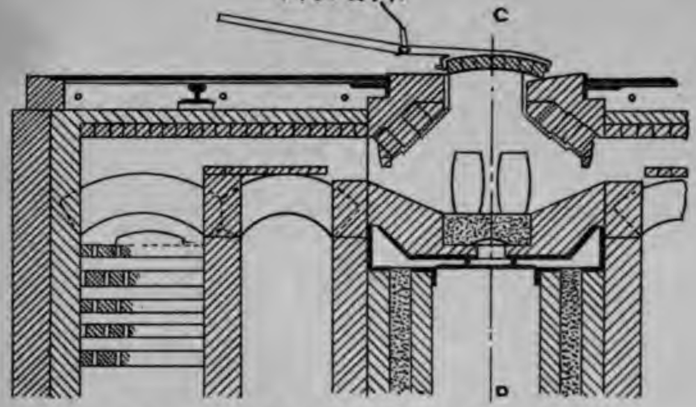
[5] "Journal Iron and Steel Institute," 1873, p. 429.

[6] "Stahl und Eisen," 1889, p. 15.

[7] "Journal Iron and Steel Institute," 1896, II, p. 161.

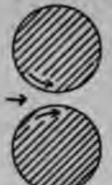
[illegible]

FIG. 277.



SECTION THROUGH AB.

FIG. 285



TWO- AND THREE-HIGH ROLLS.

FIG. 286.

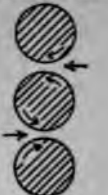
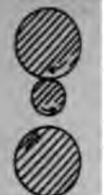
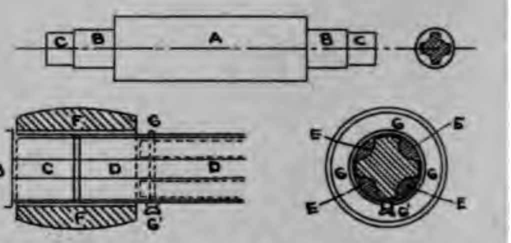


FIG. 287

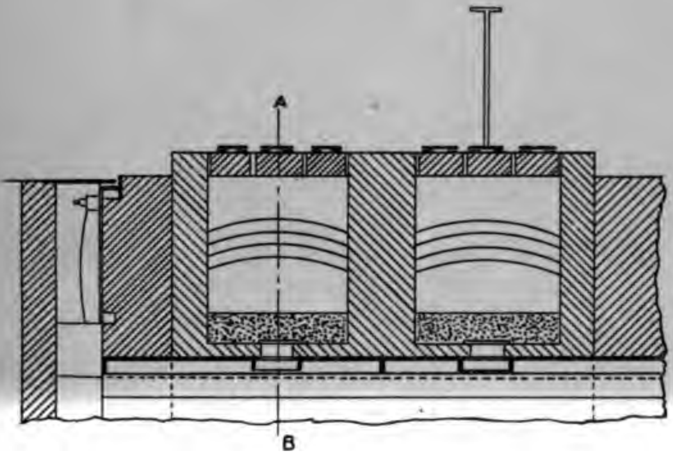


LAUTH PLATEM

FIGS 289-292.



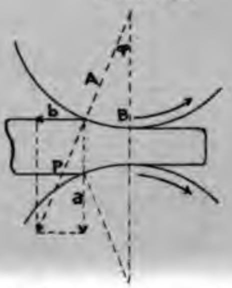
CONNECTING OF ROLLS.



SECTION THROUGH CD.

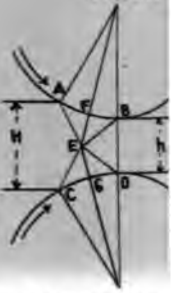
SIEMENS CRUCIBLE FURNACE.

FIG. 294



WORK OF ROLLS

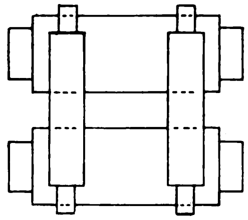
FIG 296



LENGTHENING

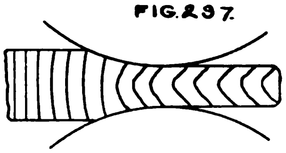
METAL BY ROLL

FIG 288.



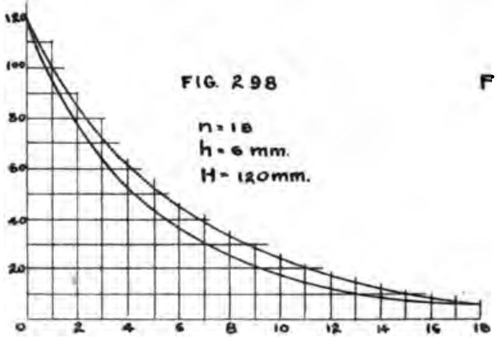
UNIVERSAL MILL.

FIG. 297.



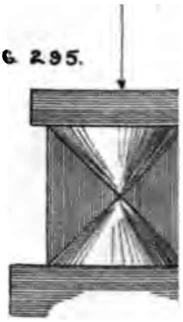
FLOW OF METAL IN ROLLING.

FIG. 298



REDUCTION OF METAL IN ROLLING.

FIG. 295.



CRUSHING

BRITTLE CYLIN

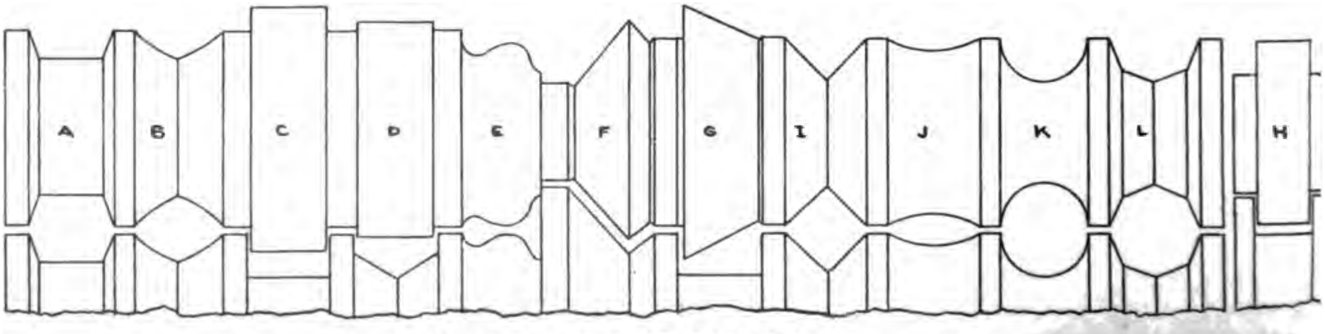


FIG. 299 FORMS OF PASSES.

(2) *Carbon Monoxide* is readily formed in liquid steel by the action of dissolved or suspended Fe_xO_y, and the escaping CO assists in the passing off of dissolved H. This explains the presence of H with CO in cavities of ingots, the former strongly predominating.[1] It has not been proved, so far, that CO is dissolved in liquid steel as is H; it has not been found fixed in the solid metal.

(3) *Hydrogen*[2] is readily dissolved by steel and alloyed with it; small alloyed amounts affect the mechanical properties. One volume of electro-deposited iron[3] may contain as many as 250 volumes of H, but industrial iron retains less than 0.5 volume H. According to Heyn (*l. c.*), steel heated to a temperature of 730–1,000° C. will absorb H, and this will diffuse through the mass. If the metal is chilled, the diffused H will remain occluded and render the metal brittle; if it is allowed to cool slowly, the H passes off and the cold metal will be found not to have been affected; if the chilled metal is reheated slowly, the H will be driven out and the metal will regain its former mechanical properties. The amount of H absorbed by melted iron depends upon the pressure under which it is and upon the chemical composition. It is known in a general way that Mn, Ni, and CO favor the absorption, that C opposes it, and that Si retards its liberation when once absorbed.

J. ROLLING OF STEEL.

Literature: Daelen-Hollenberg-Dickman, "Verhandlung zur Beförderung des Gewerbefleisses," 1869, XLIII, supplement; Beckert, "Leitfaden der Eisenhüttenkunde," Springer, Berlin, 1900, Part III; Angström, Walzen- und Kalieber-construction für Eisenwalzwerke, "Oesterreichisches Jahrbuch," 1891, XXXIX, pp. 353–423; Ledebur, "Manual," 4th ed., p. 811; Ledebur, "Lehrbuch der Mechanisch-metallurgischen Technologie," Vieweg, Brunswick, 3d ed., 1904; Brovort, "Das Kaliberieren der Walzen," Felix, Leipsic, 1903; Geuze, "Laminage du Fer et de l'Acier," Béranger, Paris, 1900; Tunner-Pearse, "Roll Turning," Van Nostrand, New York, 1869; Spencer, "Roll Turning," Spon, London, 1891, 1893; Harbord, "Steel," p. 283 and following; Kirchberg, Kalibrieren der Walzen, "Stahl und Eisen," 1903, p. 1141; "Iron Age," December 10, 1903; Risdale, The Correct Treatment of Steel, "Journal Iron and Steel Institute," 1901, II, pp. 52–103; While, Rolling of Sections, "Iron Age," March 27, 1902; Hirst, Roughing Rolls, *op. cit.*, March 16, 1892; Method of Determining Dimensions of Oval Passes, *op. cit.*, August 29, 1895; Reduction and Effect of Diameters on Bars in Rolls, *op. cit.*, September 26, 1895; Difficulties in Rolling, *op. cit.*, June 30, 1898; The Design of Roughing Rolls, *op. cit.*, March 16, 1899; The Design of Passes for Wire Rods, *op. cit.*, May 10, 1900; Angle Rolls, *op. cit.*, August 8, 22, 1901; Schwarze, Electrically Driven Rolls, "Stahl und Eisen," 1901, p. 1081.

SECTION 101. **In General.** Iron and steel are usually rolled if they are to be converted into pieces of nearly uniform cross section. Rolls are cast-iron cylinders moving in opposite directions; the friction between a pair of rolls and the metal causes the former to nip the metal and draw it in, when it will be reduced in a vertical plane to the distance between the rolls

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[1] Müller, "Stahl und Eisen," 1879, p. 494; 1881, p. 6; 1882, p. 531; 1883, p. 443.

[2] Heyn, "Stahl und Eisen," 1900, p. 837; "Metallographist," IV, p. 40; Wedding-Fischer, "Stahl und Eisen," 1903, p. 1268; "Iron and Steel Magazine," I, p. 203.

[3] Wedding-Fischer, *l. c.*

and increased in length, but not much in width. Rolls are either plane or grooved. With plane rolls, only plates can be rolled. If the metal is to have special forms, grooves are cut into the upper and lower roll. The space between two grooves of a roll or between two plane rolls is called a pass. A bar traveling through the space is said to have made a pass. As the amount of work to be done in rolling is very great, it is impossible to obtain the final reduction by a single pass; the general scheme of rolling is to have a succession of decreasing passes, which reduce the size of a bar and at the same time increase its length.

Rolls are two-high, three-high, or universal. With two-high rolls (Figure 285), the bar is fed only from one side (the roller's) and returned from the other (the catcher's) over the top roll unless the rolls are reversing. The bottom roll rests with its journals in fixed boxes; the upper roll, when movable, is balanced by steelyards and weights and adjusted by means of screws. With three-high rolls (Figure 286), the bar fed between the bottom and middle roll is returned between middle and top roll. The middle roll is either fixed and drives by gearing the upper and lower rolls (Figures 217–219), or, as in the Lauth plate mill (Figure 287), the big upper and lower roll is driven positively, while the small middle roll is driven by the friction of the other two. It is raised and lowered (see Figure 306) so as to be backed its full length alternately first against one large roll and then against the other. With a universal mill (Figure 288), the vertical as well as the horizontal rolls have a reducing action on the ingot; the vertical rolls, however, rather keep the piece to the desired width and prevent edge-cracks than effect much reduction.

The leading parts of a train of rolls have been given in Figure 173, a train being the term used for a number of rolls set up with their respective housings in a continuous line and coupled in such a manner that all shall have the same velocity. The manner of connecting rolls is shown in Figures 289–292; $a =$ body of roll, $b =$ neck, c and $d =$ star-shaped coupling (pods, wabblers), $f =$ sleeve (coupling box), with projections e fitting loosely into the recesses of the star-shaped coupling so as to permit varying the distance between the upper and lower rolls. In the table accompanying Figure 293 are given the dimensions of some of the typical rolls made by the Carnegie Steel Company. Rolls[1] are made principally of cast iron, occasionally of steel. The cast iron can be soft, medium hard, or chilled, depending upon the kind of work that is to be done. Soft rolls will be made of a tough gray iron, medium-hard rolls perhaps of mottled iron. Nicholson (*l. c.*) gives the following as an ideal composition for a chilled roll: Si, 0.650; S, 0.050; P, 0.250; Mn, 1 500; Gr. C., 2.865; C. C., 0.635. Steel rolls were formerly confined to cold-rolling soft metals; on account of their resistance to wear they have been introduced into rail mills, the rolls of which were formerly made exclusively of chilled cast iron; chilled cast-iron rolls find their principal use in sheet- and tin-plate mills.

SECTION 102. **The Work of Rolls.** If, in Figure 294, a bar be placed against a pair of moving rolls, a radial pressure P will be exerted upon it. This resolves itself into two components, the vertical $a = P \times \cos \phi$ which causes the reduction of area and the friction between roll and bar, and the horizontal $b = P \times \sin \phi$ which tends to prevent the bar from entering the rolls. As long as the friction resulting from a is greater than b, the bar will pass through

[1] Casting of Rolls: "Iron Age," April 23–30, 1903; "Iron Trade Review," July 23, 1903, July 7, 1904; "Foundry," 1903, XXII, p. 227; Sharp, "Modern Foundry Practice," Spon, London, 1900, p. 507; Beckert, "Eisenhüttenkunde," III, p. 130; Dingler, "Polytechnisches Journal," CCXCVII, p. 26; Nicholson, Chilled Rolls: "Foundry," 1899, XIV, pp. 38, 62; Winder, Failures in the Necks of Chilled Iron Rolls: "Journal Iron and Steel Institute," 1892, II, p. 176.

the rolls, and will be rolled if it is plastic, or broken if brittle. Now a and b are functions of ϕ; with $\phi = 0$ we have $b = 0$, *i. e.*, the bar will readily pass through the rolls as long as its thickness is not greater than the distance between the rolls. The angle ϕ, *i.e.*, the pressure required, increases with a thicker bar, or with rolls of a smaller diameter, as in either case b becomes larger. But the arc AB increases with the diameter of the roll, and as the friction between bar and roll is proportional to the surface, it follows that with a greater diameter of the roll there is an increase in the power to draw in the bar. This shows that *rolls of large diameter nip more readily than those of small diameter;* also that *thin bars are more readily nipped than thick bars.* Practical experience has shown that under ordinary conditions a bar will not be nipped by a smooth roll if its thickness exceeds one-tenth the diameter of the roll on entering and one-twentieth after passing the roll.

If a cube of a brittle substance be subjected in the direction of one axis to a gradually increasing pressure until it breaks, it will theoretically be found to have split into six pyramids, all of which have their apices in the centre of the cube. If the brittle substance has the form of a cylinder (say a stearin candle), there will be formed two cones (Figure 295), joined at the apices and with the bases coinciding with the pressure planes, while the material outside of the cones will have shelled off. If the brittle substance was a rectangular block instead of a cylinder, there will be formed two truncated pyramids joined at the smaller bases. If the body subjected to pressure was a plastic metal, the pressure cones or pyramids will also form as with brittle material; but instead of shelling off, the metal will flow from the apices to the bases of the cones (cups) or pyramids. The formation of these can be seen in pulling apart an iron rod in a testing machine, when one surface of fracture will have a convex, the other a concave form.

If a metal is subjected to pressure between rolls, two small cones form in the line in which the pressure is being exerted, while the rest of the metal flows out in the other two directions so that it becomes flattened and lengthened. The amount of flattening is independent of the width of the original piece, but dependent on its thickness and increases with it. In rolling plate between smooth rolls or a rod in an open pass (see Section 104), the edges become uneven and ragged and have to be trimmed later on. If in a closed pass there is not sufficient room to allow for the flattening out of the piece, considerable lateral pressure will be exerted by it on the collar (flange) unless special precautions are taken to round off the form of the groove. The lengthening of the metal in rolling is represented in Figure 296. Let H and h be the thickness of the bar before entering and after leaving the rolls. In passing from H to h, the bar is compressed by the arcs AB and CD of the rolls. The radial pressure of the rolls causes the formation of pressure cones ABE and CDE. If the rolls move through the arc $FB = GD$, the bar of the section $BFEGD$ will be drawn through the rolls and leave them at BD. As a rectangle of the length BF and the height h is smaller than the surface enclosed by $BFEGD$, its length must become larger than BF, or the length of a bar, after being rolled, is greater than the arc of the roll that did the work. It can be proved that the difference in velocity of a bar entering and leaving rolls is the greater the smaller the diameter of the rolls, or, *rolls of small diameter increase the length of a bar more quickly than those of large diameter.* Therefore in rolling small rods (which cool very quickly) the diameter of the rolls will be made as small as they can without danger of breaking.

The flow of metal[1] that takes place in rolling is shown by the curved lines in Figure 297. It can be easily demonstrated[2] by boring holes into a bar in the direction in which the rolls exert their pressure, inserting pieces of wire, and planing down the bar after rolling. With cold rolling, the curvature of the wire will be greater along the surface of the bar; with hot rolling, greater along the centre.

The power required for rolling depends on the friction between rolls and bar and on the resistance of the metal to the flow of its molecules. It grows with the square root of the radius of the roll and that of the thickness of the bar; hence the diameter of a roll is made as small as is consistent with the safety.

SECTION 103. **Draught of Rolls.** The reduction of a metal in rolling may not exceed a certain amount without the metal being torn. The greater the reduction the greater the saving in labor (because fewer passes are needed), and to some extent in power (because the metal is hotter and hence softer). The permissible reduction depends on the temperature and resistance of the piece, the power of the rolls, and the ratio of size of rolls and of piece. Wrought iron or low-carbon steel, being usually hotter and always softer than high-carbon steel, can stand a greater reduction; it will also be more drawn out and less flattened. Rolls of large diameter can exert a greater pressure and nip better than those of small diameter; but the larger the diameter and the slower the speed the more will they flatten the piece and the less will they lengthen it. Other things (temperature, hardness, section of piece) being equal, the resistance to rolling will be smallest if the reduction is distributed evenly over a number of passes. If H is the section of the original piece; $H_1 = xH$, the section after the first pass; $H_2 = x^2H$, after the second pass; $H_n = x^nH = h$, after the nth pass; then $x = \sqrt[n]{\frac{h}{H}}$ is the reduction factor. Supposing that a plate 120 millimeters in thickness is to be reduced to 6 millimeters in eighteen passes with a constant reduction factor, then $x = \sqrt[18]{\frac{6}{120}} = 0.84668$. In Figure 298, the upper curve represents graphically this reduction. On account of the increasing hardness of the piece in rolling, the lower curve (by Gresser) with an increasing reduction factor is more satisfactory than the upper one with a uniform factor.

With grooved rolls the matter becomes more complicated, as the grooves being turned into a roll will weaken it more at the centre than at the ends. Therefore the larger grooves should be put in the ends and the smaller ones near the centre. If the curve of reduction has been drawn, it will have to be distributed over the number of pairs of rolls that are to be used, and in each pair the reduction in the middle will have to be smaller than at the ends. This will give the reduction curve the form of a serpentine line.

SECTION 104. **Passes of Rolls.** If a piece is to be rolled into a definite shape, at least one roll of the pair will have to be grooved. The combination of the groove of one roll with the groove, the projection, or the plain surface of another roll forms a pass. Passes have various forms, as shown in Figure 299. Passes A and B are open. The grooves are evenly divided

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[1] Roberts-Austen, "An Introduction to the Study of Metallurgy," London, 1902, pp. 52, 56; Cuthill, "Colliery Guardian," January 22, 1897; Tresca, On the Flow of Solids: "English Institute of Mechanical Engineers," 1867, p. 114; Loss, Flow of Metal in Punching and Shearing: "Journal Franklin Institute," 1901, CLI, p. 456; "Iron Age," June 20, 1901. Heyn, Photomicrograph of punched metal: "Stahl und Eisen," 1889, p. 770, Plate XVII, Figure 39.

[2] Hollenberg, "Stahl und Eisen," 1883, p. 121.

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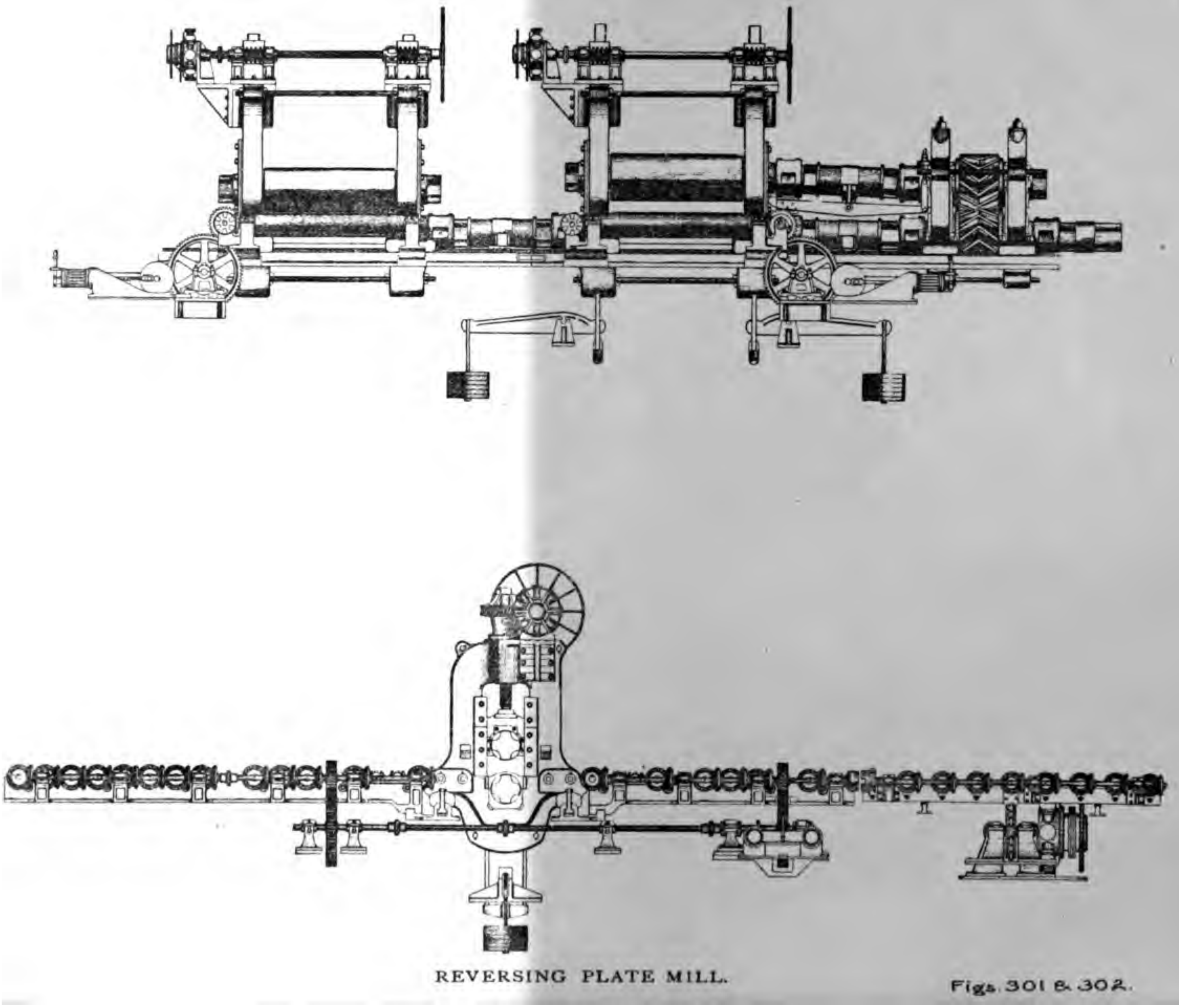
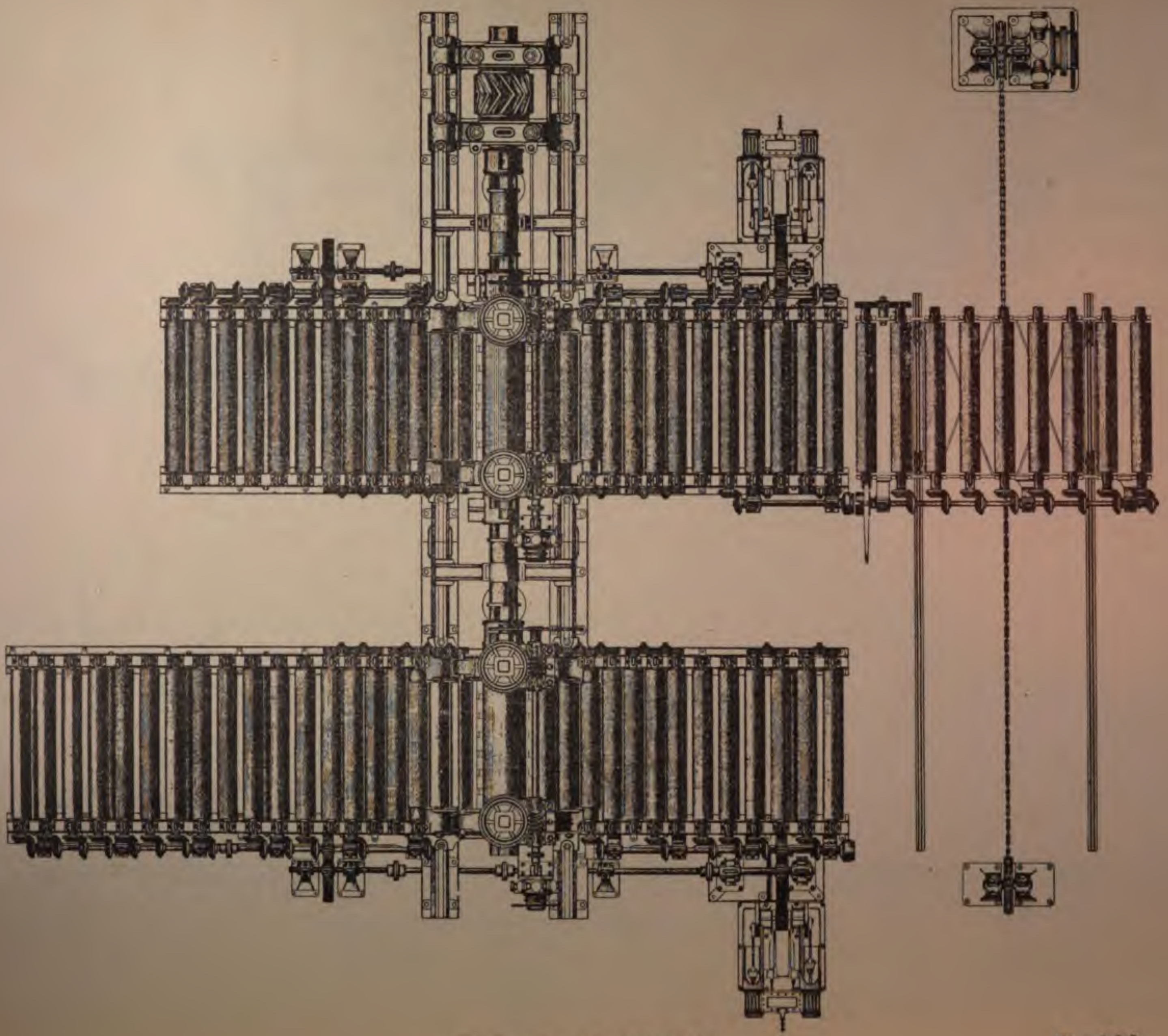
ASTOR, LENOX AND
TILDEN FOUNDATIONS.



REVERSING PLATE MILL. Figs. 301 & 302.



REVERSING PLATE MILL.

Fig. 300.

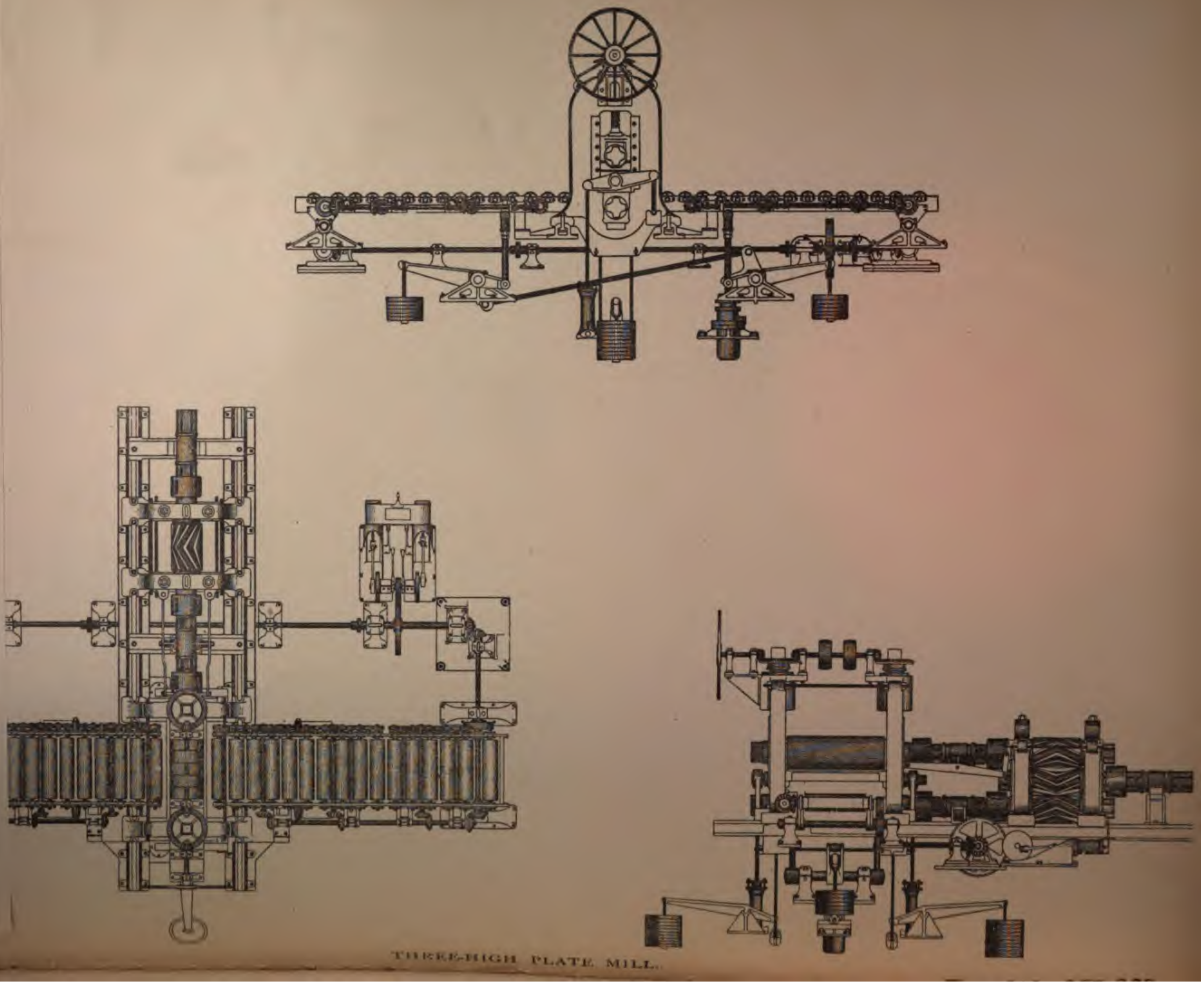
between top and bottom roll; *A* is a box pass and *B* a Gothic pass. Passes *C* and *D* are closed passes. The grooves *C* and *D* are cut deep enough into the lower roll to receive the projection of the upper one, which closes the pass. By thus sinking the whole pass into the lower roll and allowing the collar on either side to project into the groove of the upper roll, the height of the pass can be changed by raising or lowering the upper roll without opening the pass. Passes *E* and *F* are half-open passes. They are closed on one side and open at the other. As an open-pass roll is less deeply cut than a closed one, it is stronger, but it cannot shape the piece so accurately, as there is greater tendency to "finning," *i. e.*, the metal squeezing out between the crevices of the upper and lower rolls. This is because the metal flows outward sidewise more readily from the middle of the side than from the corner. The remedy for the evil is to roll the fin back into the piece before it grows too large by an edging pass, *i. e.*, turning the piece 90° so as to bring the fin against the solid body of the roll. Finning is prevented by making each successive pass enough wider than the preceding one so that the fin cannot reach the crevice between the rolls. In order to avoid this widening of passes, the piece is turned 90° when the former width becomes now the depth (thickness), and the flattening out is begun again. With Gothic or diamond passes the piece must be turned 90° at each pass, therefore the successive passes must be so proportioned that the depth (height) of one pass shall be a little smaller than the width of the next. Pass *G* is an unbalanced pass. It deflects the piece horizontally and increases the friction of the rolls in their bearings. The piece on leaving the rolls tends to turn toward the least reduced side, and if it is straightened by a guard it is liable to tear. According to the shape of the passes, there are distinguished Gothic passes (*A*), flat passes (*C* and *H*), diamond passes (*I*), oval passes (*J*), round passes (*K*), polygonal passes (*L*), and special shapes (rails, angle irons, etc.). According to the kind of work that the rolls are intended to do, there are roughing and finishing passes, welding, drawing, shaping, flatting, edging, adjusting passes, etc., and the rolls are called roughing rolls, finishing rolls, etc.

In balanced passes the piece ought to emerge straight from the rolls; it may, however, be deflected vertically and horizontally. Vertical deflection is caused (1) by the friction of the collar against the piece, when the piece tends to adhere to the deeper-grooved roll; (2) by one roll having a greater circumferential speed than the other, when the piece follows the slower roll; (3) by the diameter of the groove of the top roll being greater than the diameter of the groove of the lower roll, when the top surface of the piece is drawn out more than the bottom surface, and the bottom surface curls downward; (4) by the piece being unequally heated, when it tends to bend toward the hotter side. Horizontal deflections are caused by unbalanced rolls and by unequal heating. Corrections are made by guides.

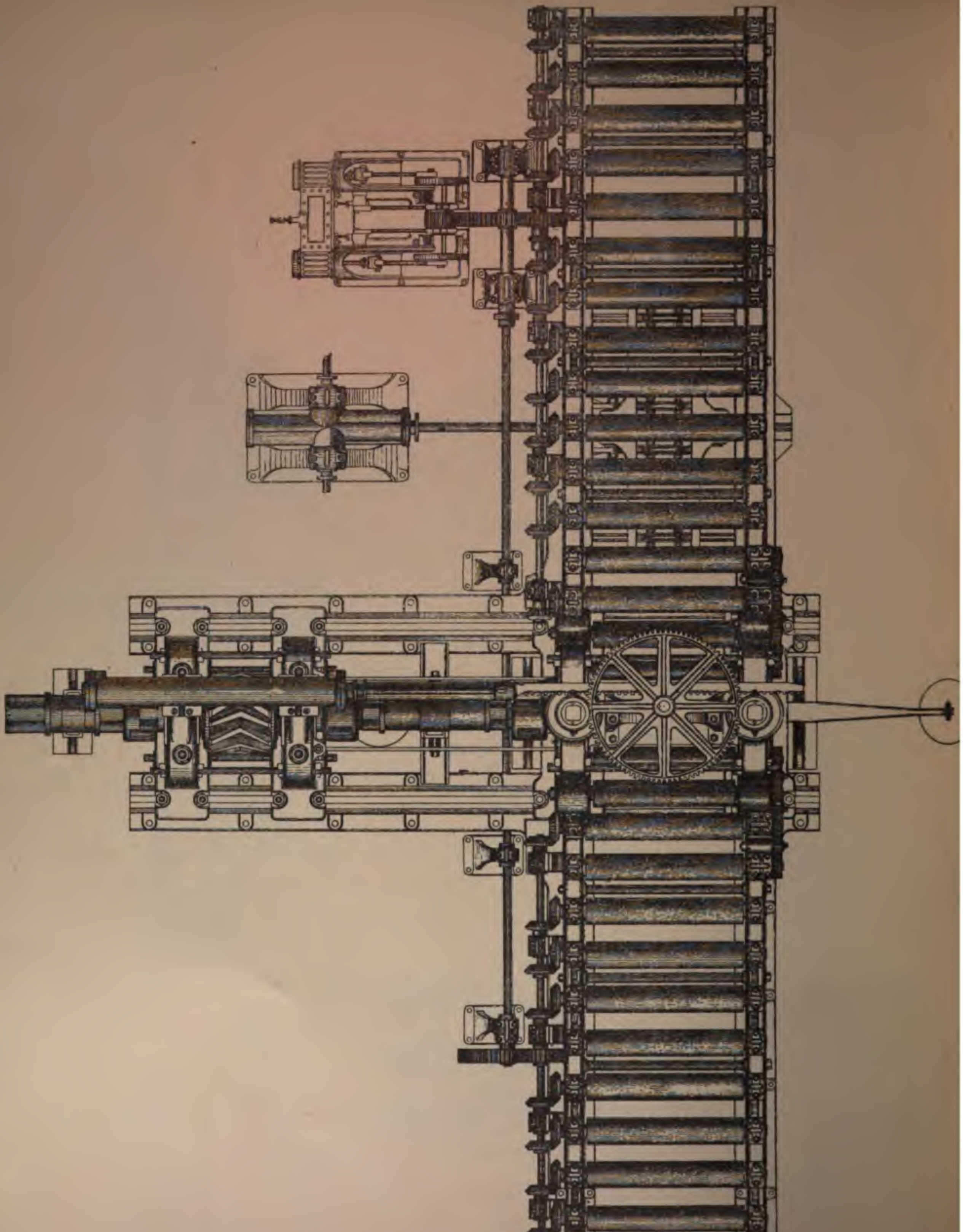
SECTION 105. **Examples of Rolling Mills.**

(1) *Three-high Non-reversing Blooming Mill.* See Figures 217–219.

(2) *Two-high Reversing Plate Mill* (Figures 300–302). The illustrations represent two pairs of plate rolls driven by one engine. One pair serves as roughing, the other as finishing rolls. Each pair of rolls has a stationary feeding table with live rollers. The plate, reduced to the required size in the roughing rolls, is delivered by the live rolls on to a pair of rails, over which it is pulled mechanically on to the feed table of the finishing rolls. The working of the rolls differs from that of the blooming mill previously described, in having the upper rolls adjustable by means of screws; further, in the roughing mill the upper roll is balanced or pressed



THREE-HIGH PLATE MILL.



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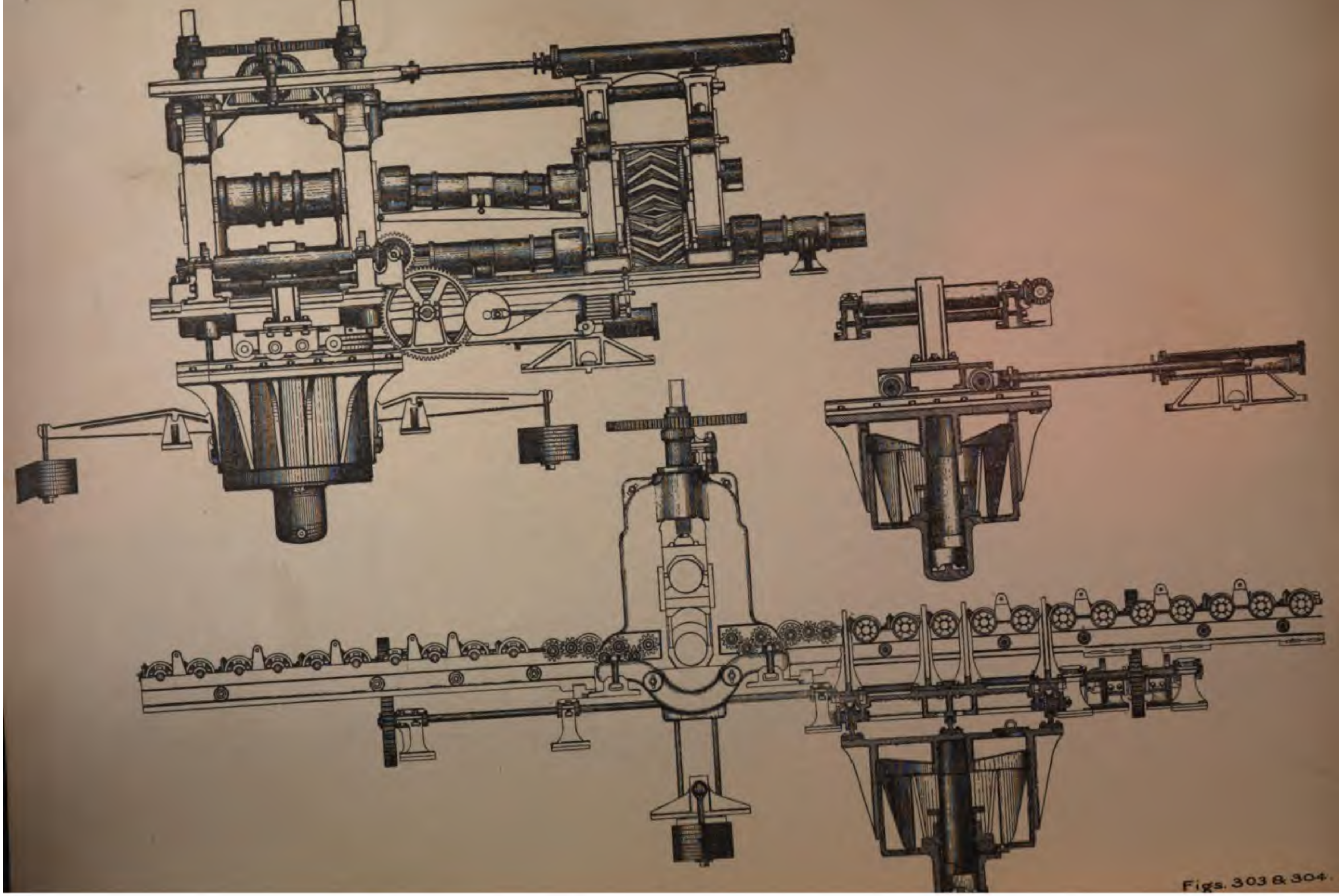

Figs. 303 & 304.

against the shoes of its adjusting screws by stay-rod, lever, and counterweight. In the finishing roll there is no such balancing; the plate, upon entering, raises the upper roll, pressing it against the shoes of the adjusting screws.

(3) *Two-high Reversing Blooming Mill* (Figures 303–305). This is a very common form of mill, because it permits roughing down ingots with one set of rolls to all sizes of squares and flats within the range for which the mill was constructed. The manner of supporting, driving, and adjusting of rolls and of manipulating the ingot is shown clearly in the illustrations. The mill of the Chester Rolling Mills, Chester, Pennsylvania,[1] has a two-high 32-inch reversing blooming mill; the rolls are 26 inches in diameter; the engine has a 30-inch cylinder and a 48-inch stroke.[2]

(4) *The Three-high* (*Lauth*) *Plate Mill* (Figures 306–308). This is the standard type of large-size plate mill in use in this country. The characteristics are a fixed large-size bottom roll, an adjustable large-size balanced top roll, and a small-size middle roll; the manipulation of the plate by means of lifting tables with live rollers is similar to that in the three-high blooming mill (Figures 217–219), excepting that there are no ingot manipulators. The raising and lowering of the middle roll by means of lever, toggle, and piston rod is clearly shown in Figure 306. The plate mill of the Illinois Steel Company[3] has two stands of rolls of this type driven by a Porter-Allen engine with 54-inch steam cylinder and 66-inch stroke, making sixty revolutions per minute. The first (inner) stand next to the pinions is 90 inches long, the top and bottom rolls are 34 inches, the centre roll 18 inches in diameter, all of chilled iron. The second (outer) stand has rolls 132 inches long; the top and bottom rolls are of steel, and 34 inches in diameter; the middle roll is 21 inches in diameter and of chilled iron. The mill admits ingots 24 inches thick, and the table will handle ingots weighing 7 or 8 tons.

(5) *The Universal Mill* is shown in Figures 309–311. It serves principally for rolling plates of special sizes which are long, comparatively narrow, and thin, say 8–42 inches wide and 0.25 inch thick. The plates are finished with square and straight edges, so as not to require any shearing. As in the ordinary two-high plate rolling mill, the top horizontal roll is balanced by stay rods, levers, and counterweight, and adjusted by screws, while the bottom roll is journaled in fixed bearings. The vertical rolls are driven by means of bevel gearing, set in motion through a shaft having at its opposite end a spur-gear, meshing with another keyed to the shaft which drives the top horizontal roll. Thus in the rolls represented by the figures the speed of the vertical rolls is guided by that of the horizontal rolls, which is not always the case.[4] The adjustment of the vertical rolls to enlarge or narrow the space between them is not clearly shown in the figures. Probably the short shaft, with the bevel gear at the right end, is hollow, and has one or more grooves into which the continuation of the solid shaft beyond its bevel gear fits. This would enable the vertical rolls to separate and at the same time to be driven from one main shafting. The bearings of the vertical shafts are probably carried in guides and adjusted by screws having gearing at either end (Figure 309) which meshes with a central spur wheel. By means of a lever (Figure 310) one or the other of the two sets of gearing will be

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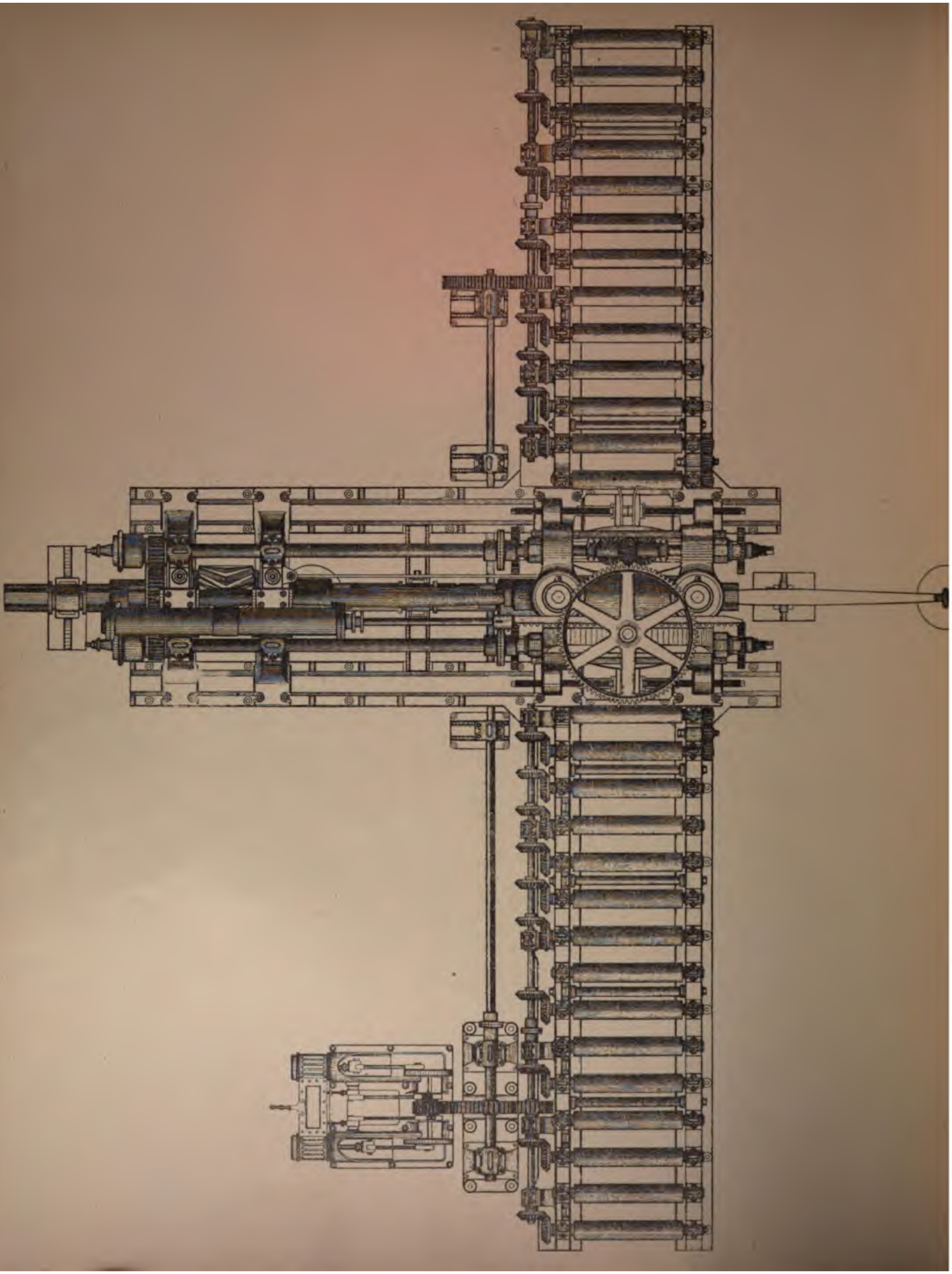
[1] "Iron Age," April 10, 1890.

[2] See also "Journal Iron and Steel Institute," 1895, II, Plate XIX, facing p. 144.

[3] "Iron Age," July 25, 1895, December 10, 1896.

[4] See below, Pennsylvania Steel Company.



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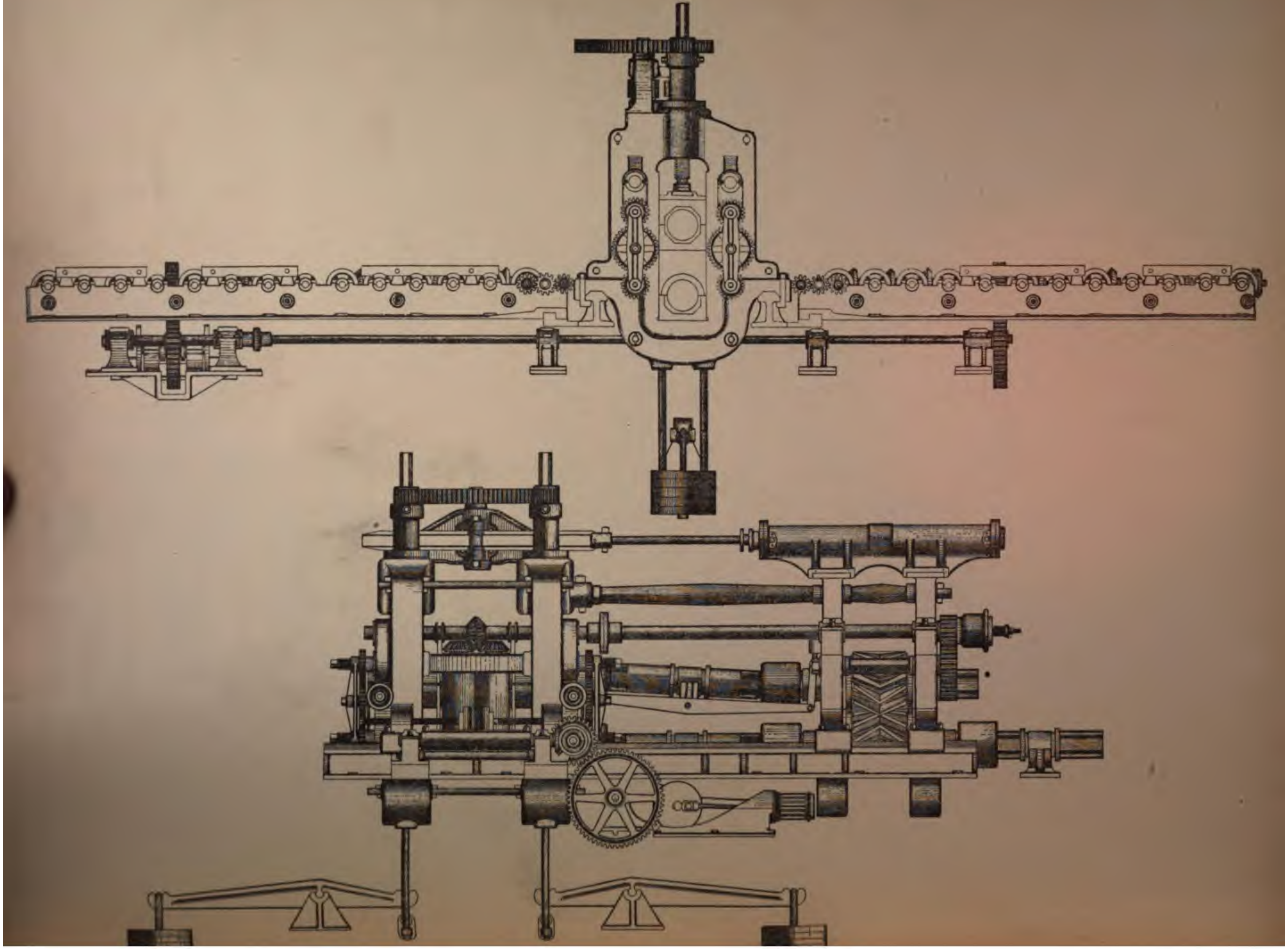

set in motion and the vertical rolls brought closer together or farther apart. The side guides (Figures 310–311) follow the lateral motion of the vertical rolls, being directed by the fingers (eccentrics) (Figure 310).

The universal Slabbing Mill of the Pennsylvania Steel Company[1] is somewhat different in detail from that given in Figures 309–311; it has horizontal rolls 26 inches in diameter and vertical rolls 20 inches in diameter. It usually handles ingots 24 x 32 inches (can handle 32 x 52 inches, does handle 32 x 40 inches and 24 x 44 inches). The capacity is 1,000 tons per day.

(6) *References to Rolling Mills. In General:* Durfee, Early Use of Rolls, "Cassier's Magazine," 1899, XV, p. 478; Hunt, Evolution of American Rolling Mills, "Iron Age," November 19, 28, 1891; "Engineering and Mining Journal," 1891, LII, pp. 590, 616; Garrett, Rolling Mill History in the United States, "Iron Trade Review," November 14, 1901; Brown, Evolution of the Rolling Mill, "Iron Trade Review," August 27, 1903; Holley, American Rolling Mills, "Journal Iron and Steel Institute," 1874, I, p. 348; Editor, New American Rolling Mills, "Stahl und Eisen," 1897, pp. 136, 181, 214; Eyermann, American Rolling Mills, *op. cit.*, 1900, p. 730; 1901, pp. 220, 295; Langheinrich, American Rolling Mills, *op. cit.*, 1901, pp. 1220, 1294. *Blooming Mills:* Crocker, Development of American Blooming Mills, "American Manufacturer," September 14, October 16, 1897; Sharon Steel Company, Two-high Blooming Mill, "Iron Age," July 4, 1901. *Billet Mills:* New Republic Billet Mill, "Iron Age," October 2, 1892; Duquesne, Merchant Bar Mill, *op. cit.*, and "Iron Trade Review," January 8, 1903. *Plate Mills:* Crowe, American Plate Mills, "Iron Trade Review," July 10, 1902; Garrett, Plate Mills, *op. cit.*, April 17, 1902; Thurlow Plate Mill, "Iron Age," March 31, 1892; Cambria Steel Company, 34-inch Slabbing Mill, "Iron Trade Review," July 25, 1901; Illies, Sheet Mill, Carnegie Steel Company, "Iron Age," May 3, 1903; Bray, Two-high Sheet Mill, "Iron Age," May 9, 1903. *Universal Mills:* York Universal Beam Mill, *op. cit.*, May 15, 1890; Pennsylvania Steel Company, Universal Slabbing Mill, "Iron Age," April 11, 1895; Homestead, Universal Mill, *op. cit.*, November 15, 1898, December 27, 1900. *Rod and Wire Mills:* Hunt, Rod Mills, "Iron Age," March 13, 1892; Daniels, Rod Mills, *op. cit.*, August 10, 17, 31, 1893; Garrett, Rod Mills, *op. cit.*, January 2, 1896, February 21, 1901; Baakes, Rod Mills, "Stahl und Eisen," 1900, p. 65; Morgan, Continuous Rod Mills, "Iron Age," September 12, 1901; Rod Mill with Open-hearth Plant, "Iron Trade Review," March 5, 1903; Garrett, American Wire Mills, "Stahl und Eisen," 1902, p. 545. *Mannesmann Tube Mills:* H. Leobner, "Ueber das Mannesmannsche Röhrenwalzverfahren," Hamburg, 1897; "Iron Age," July 24, 1900, January 5, 1901; "Engineering and Mining Journal," June 14, 1890; "Transactions American Institute of Mining Engineers," XIX, p. 384. *Heating Furnaces:* Beard, Annealing Sheet Iron, "Iron Age," January 4, 11, 1894; Swindell, *op. cit.*, July 10, 1902, also mills quoted. *Shears:* See mills quoted; further, Aiken, "Iron Age," July 24, 1890, November 27, 1890; Pietzka, *op. cit.*, December 15, 1898; Morgan-Allen, *op. cit.*, February 27, 1896; Jones, *op. cit.*, July 10, 1902. *Saws:* "Iron Age," December 25, 1890. *Presses:* Twedell, "Stahl und Eisen," 1894, p. 900; "English Institute of Civil Engineers," February 20, 1894; "Iron Age," January 2, 1890, October 29, 1891, August 25, 1892, September 6, 1894, November 1, 1894; Daelen, "Journal Iron and Steel Institute," 1901, I, p. 146; Mercader, Hollow-pressed Axles, *op. cit.*, 1903, I, p. 95. *Hammers:* "Manuals" of Beckert,

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[1] "Iron Age," April 11, 1895.

Harbord, Ledebur; Latrobe, Pennsylvania, "Iron Age," March 27, 1890; Axle Works, Carnegie Steel Company, *op. cit.*, June 13, 1901.

K. HEAT TREATMENT OF STEEL — HARDENING, ANNEALING, AND TEMPERING.

Literature: Ede, "The Management of Steel," 7th ed., London, 1901; Reiser, "Das Härten des Stahles in Theorie und Praxis," 3d ed., Leipsic, 1900; Thallner-Brannt, "Tool Steel," Philadelphia, 1902; Woodworth, "Hardening, Tempering, Annealing, and Forging of Steel," New York, 1903; Howe, "Steel," p. 22; Harbord, "Steel," p. 640.

SECTION 106. **Heat Treatment in General.** If steel with 0.3–1.3 per cent. C (below 0.3 per cent. C hardening is difficult to detect; above 1.3 per cent. C steel becomes softer when chilled, Osmond) be heated above the critical point, *i.e.*, above 700° C., and then cooled suddenly by plunging into water, oil, or some other medium, it will become *hardened* and brittle, as most of the carbon is present in the form of hardening carbon. The general effect[1] of hardening, beside increasing the hardness of steel, is to raise the elastic limit, diminish the ductility and specific gravity, and, if the cooling be not extremely sudden, to raise the tensile strength. If the steel is cooled down slowly from above the critical range, the hardening carbon is changed into cement carbon, and the steel has been *annealed;* it has become soft and tough. Internal stresses caused by cold working have been removed. If the effects of hardening are only partly removed, the steel is said to have been *tempered.* The following diagram by Sauveur[2] (Figure 312) represents the changes that unhardened and hardened steels undergo when heated to different temperatures and then cooled slowly or quickly (quenched). Of the two heavy horizontal lines *V* and *W*,[3] the line *V* represents the temperature (point of recalescence) at which hardening carbon is changed to cement carbon during slow cooling, and the line *W* the temperature at which cement carbon changes into hardening carbon during reheating. *W* is generally some 30° C. higher than *V*, leaving between the two a critical range of about 25° C.

If a steel is heated to temperatures *below V* and then cooled down, its structure remains unchanged; the higher within the limit of *V* the temperature was to which the steel had been brought, the more will hardening carbon have been changed to cement carbon. *At V* all of it is changed, and the softer will have become the temper. An exception to this general statement is formed by very soft steel (0.01 per cent. C), which[4] by prolonged heating at 600–750° C. (below *W*) becomes coarsely crystalline.

When steel is heated *to W* all previous crystallization disappears and is replaced by the finest structure which the metal is capable of assuming. If now it is allowed to cool slowly, it retains this fine structure; if quenched, it becomes fully hardened.

When steel is heated *above W* and allowed to cool slowly it crystallizes again, the crystals increasing in size until *V* is reached. The higher the temperature above *W* and the slower the cooling, the larger the grains. If not too much overheated, steel can have its desired grain

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[1] Howe, "Steel," p. 18.

[2] "Journal Iron and Steel Institute," 1899, II, p. 195; "The Metallographist," II, p. 264.

[3] Brinell, "Stahl und Eisen," 1885, p. 611; "Journal Iron and Steel Institute," 1886, I, p. 365.

[4] Head, "Journal Iron and Steel Institute," 1898, I, p. 145, II, p. 137; "Metallographist," I, p. 289, II, p. 85.

restored if it is reheated to the required temperature above *W* and kept there for some time to allow the grains to assume proper form. If the steel has been much overheated (burnt, rough tears, cracks at surface and corners), it is good only for the scrap heap. If the steel heated above *W* is quenched when its temperature has reached *W*, it will be hardened and retain the coarsely crystalline structure due to the cooling down to *W*; if cooled directly after it has attained the desired temperature above *W*, the degree of hardening will remain the same, but the grain will be coarser.

Figure 313 represents a photomicrograph of burnt steel; the crystals are separated from one another, and the continuity of the mass is broken. Stansfield[1] summarizes his investigations on burnt steel by saying that when steel is heated to a temperature at which it begins to soften (see Figure 18), "a liquid begins to form which is more highly carburized than the remainder of the steel, and this accumulates at first in the intercrystalline joints, and later in large quantities, near the surface most exposed to heat. The formation of gas (due to the action of O from scale on the C of steel) at this point produces blisters and punctures in the skin, through which the liquid portion is frequently forced, leaving a cavity which becomes oxidized." See also Heyn[2], Stead,[3] Ridsdale,[4] Göransson.[5]

In order to show the changes in the grain of steel obtained by heating, Metcalf[6] nicked a steel rod (1 x $\frac{1}{2}$ inch and 6–8 inches long and brought to a low red heat) every $\frac{1}{2}$–$\frac{3}{4}$ inch, making eight to nine notches, plunged one end (No. 1) in a white-hot fire, and heated it until it scintillated, while the other (No. 8), being out of reach of the fire, became barely red-hot; he then quenched it in cold water, kept it there until cold, and then fractured it at the notches. The changes the steel underwent are recorded in the subjoined table and the fractures are shown in Figure 314.

No.	Hardness.	Fracture.
1	Scratches glass	Coarse, yellowish cast; very lustrous.
2	Excessively hard	Coarse, not quite so yellowish as No. 1; very lustrous.
3	Excessively hard	Finer than Nos. 1 and 2, coarser than No. 8; fiery lustre.
4	Excessively hard	Resembles No. 3 in color and lustre, but finer grain; coarser than No. 8
5	Well hardened	Grain same as No. 8, but fiery lustre.
6	Well hardened	Grain much finer than No. 8; no fiery lustre; hard and strong (refined by hardening).
7	Hard enough for tap-teeth	Refined and hard on corners and edges and rather strong, and not quite so hard in centre. Right heat for milling tools; teeth hard, no danger of breaking.
8	Not hardened	Original grain of bar.

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[1] "Journal Iron and Steel Institute," 1903, II, p. 448.

[2] *Op. cit.*, 1902, II, p. 73.

[3] *Op. cit.*, 1903, II, p. 128.

[4] *Op. cit.*, 1899, II, p. 113.

[5] "Transactions American Institute of Mining Engineers," XXXIII, p. 107.

[6] "Metallurgical Review," 1877–78, I, p. 245.

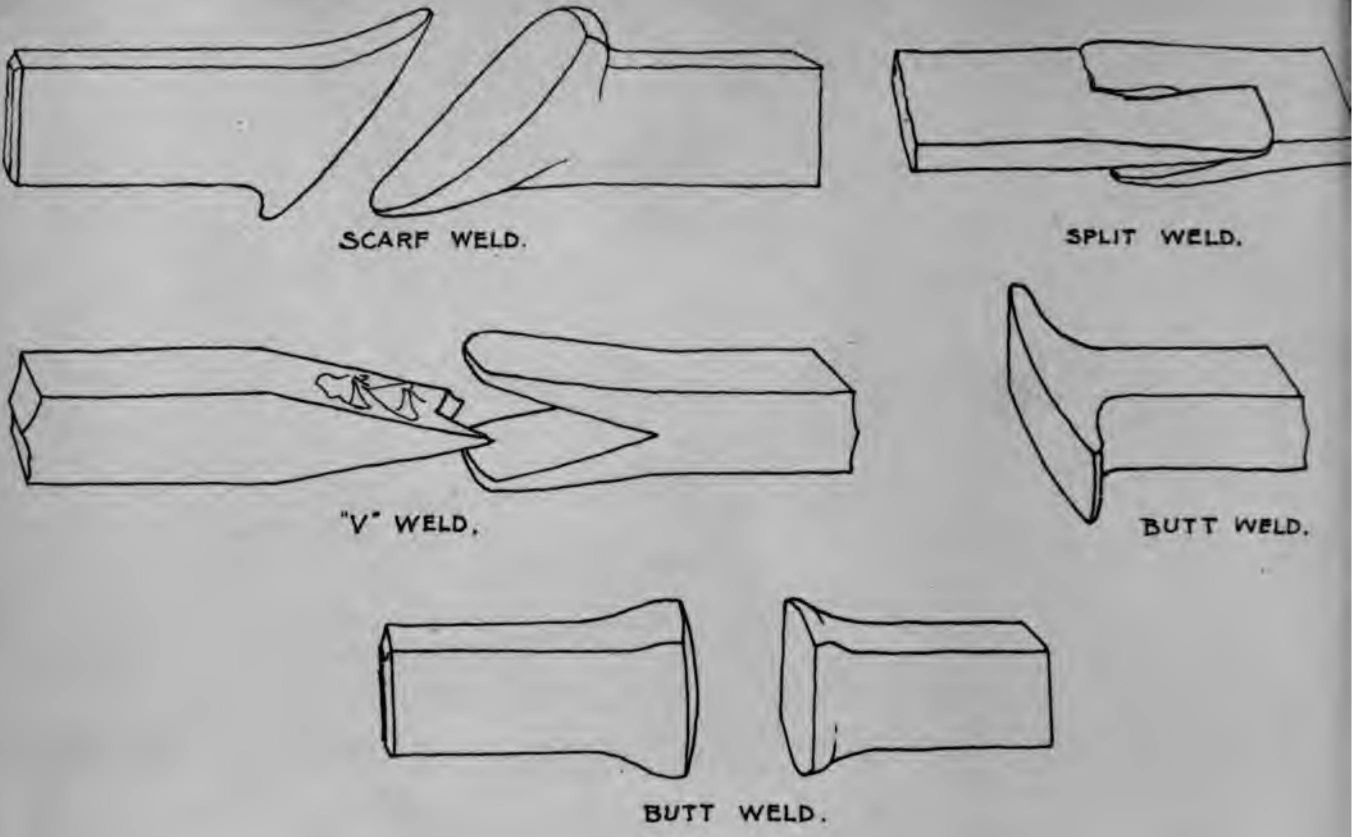
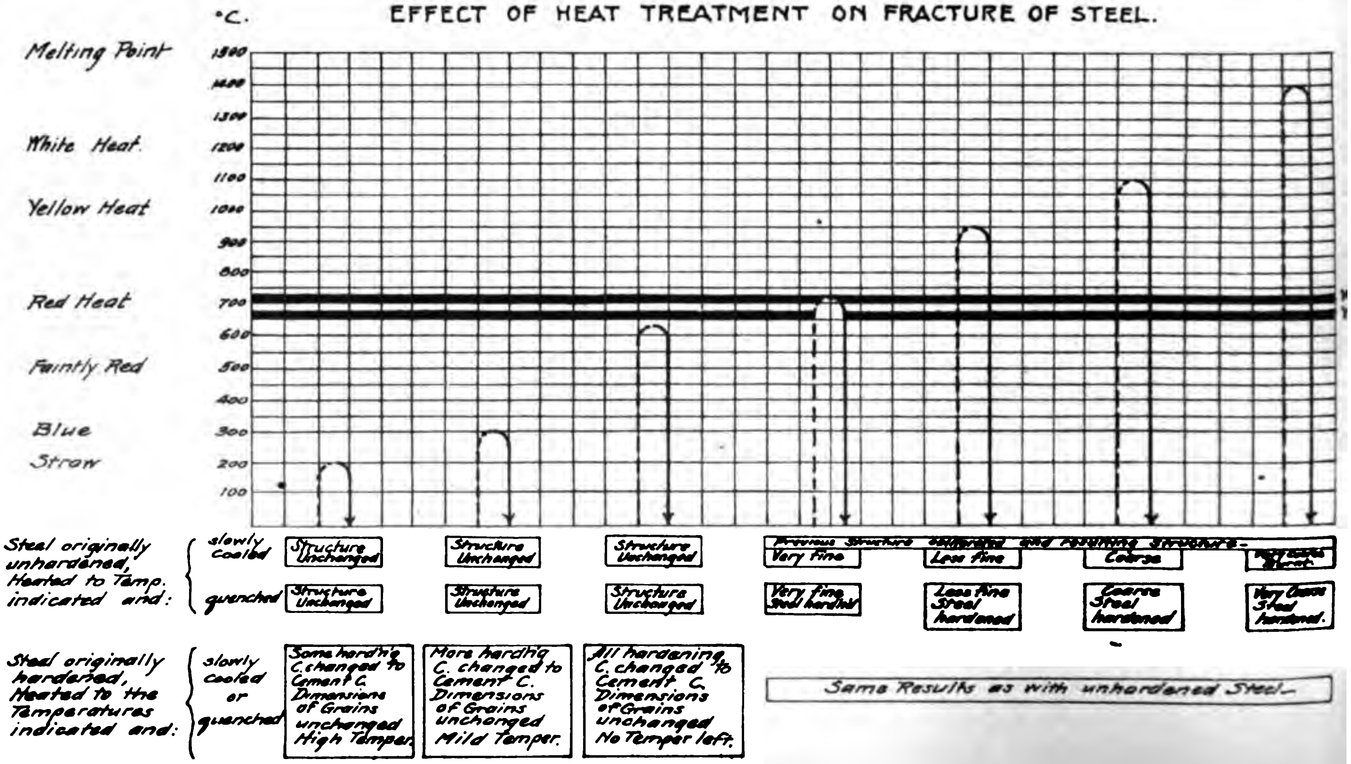


FIG. 312.

EFFECT OF HEAT TREATMENT ON FRACTURE OF STEEL.



In bringing bars to high temperatures and then quenching them, they are liable to crack down the middle as far as No. 6. In the figure, No. 4 shows the crack. A temperature high enough to open the grain makes the steel brittle. A temperature high enough to harden the steel but not to open the grain (No.6) makes the steel stronger (than No. 8). A temperature high enough to harden and refine the edges and corners alone (No. 7) is good for taps. Langley[1] determined the specific gravities of six of the above eight bars with the following results: —

Per cent. C.	0.529	0.649	0.841	0.871	1.005	1.079
Specific gravities	7.844	7.824	7.829	7.825	7.826	7.825
	7.831	7.806	7.812	7.790	7.812	7.811
	7.826	7.849	7.808	7.773	7.789	7.798
	7.823	7.830	7.780	7.758	7.755	7.769
	7.814	7.811	7.784	7.755	7.749	7.741
	7.818	7.791	7.789	7.752	7.744	7.690

They show that in hardening, the specific gravity decreases the more the higher the percentage of carbon and the higher the temperature to which the steel had been heated. In 1885, Brinell[2] studied the changes in grain that take place upon heating and cooling of medium-C steels and reproduced the fractures obtained. Morse[3] made a micrographical study of medium-carbon steels when subjected to heat treatment. Champion[4] studied the effects of heat treatment of steels with C, 0.1–0.5 per cent. Boynton[5] heated steel in a manner similar to Metcalf in his work on troostite and investigated the changes micrographically.

As heating to *W* gives steel the finest grain it is capable of assuming, which is equivalent to its highest degree of ductility and toughness, it would seem desirable to heat every finished product to *W* and allow it to cool slowly. As this is often not possible, the piece ought to be finished as near *V* as possible, as while it is being worked it does not crystallize. If finished above *V* and then allowed to cool slowly, there will be coarse grains; if finished below *V*, the cold working distorts the grains, and the more so the lower the temperature. This distortion can only be removed by heating again to *W* (see page 156, Finishing Temperature of Rails).

SECTION 107. **Hardening, Annealing, and Tempering.** *Hardening.* The hardness of steel varies with the percentage of C, within certain limits with the temperature, and with the rapidity of cooling. If it is brought to too high a temperature and quenched from this temperature, it becomes too brittle; if the temperature was too low before quenching, the steel will be insufficiently hardened. Brinell[6] found that in hardening steel it was essential to heat

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[1] "American Chemist," November, 1876.

[2] "Stahl und Eisen," 1885, p. 661.

[3] "Transactions American Institute of Mining Engineers," XXIX, p. 729.

[4] "Journal Iron and Steel Institute," 1900, I, p. 408; II, p. 497; 1903, I, p. 378.

[5] *Op. cit.*, May, 1904.

[6] *Op. cit.*, 1901, I, p. 273.

to 750° C. Steel with 0.7 per cent. C heated to 750° C. and quenched was three times as hard as when heated to only 700° C. The hardening capacity of steel increases with the C to 0.45 per cent., remains constant to 0.9 per cent., and then again decreases. In 1901, Wahlberg[1] published Brinell's "Method of Determining Hardness and Other Properties of Iron and Steel." This paper contains, beside the ball-testing method, the results of the extended researches of the effects of heating and cooling (slowly in air and quickly in water, oil, and lead) upon the physical and mechanical properties. The paper has been in part abstracted by Harbord,[2] but must be studied in the original. Arnold-McWilliam[3] have studied the microstructure of hardened steel.

Annealing. In annealing steel, beside removing the hardness and brittleness caused by hardening and the internal stresses from cold working, the aim will be to remove the coarse crystallization caused by heating above *W*, and thus bring the metal to the best physical state, when it will show the greatest ductility combined with the highest elastic limit. Steel can be completely annealed and brought to the highest degree of softness and toughness it is capable of by heating to *W*, cooling sufficiently slowly down to *V* to allow all hardening C to change into cement C, and then down to ordinary temperature. To insure slow and even cooling, the metal is often buried in charcoal.

Tempering. In tempering, the aim is to modify or remove partly only the effects of hardening to meet different requirements. The hardened steel is heated in a reducing flame to the required temperature at which it is to be quenched. Small pieces are heated, as a rule, in a blacksmith's forge, larger pieces in a reverberatory furnace, being transferred from the cooler to the hotter part. In drills, where only the edge or point is hardened and tempered, the still hot shank heats the edge or point by conduction. The temperature to which the steel is being heated is indicated with sufficient accuracy for practical purposes by the color which a polished part of the surface of the hardened steel assumes by becoming coated with a film of oxide. The subjoined table[4] gives the appearance of the films. They grow darker and darker with increase of temperature and time of exposure until a dark blue or a red just visible in the dark appears. When the steel has reached the desired color, it is quenched.

A high-carbon (hard) steel when tempered to a certain color will, of course, remain harder than low-carbon (soft) steel treated in the same manner. With a high-carbon steel a low quenching temperature gives the greatest strength; with a low-carbon steel, a high quenching temperature. The media for chilling steel arranged in the order of their activity are: mercury, brine, water, cotton-seed oil, tallow, coal tar.

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[1] "Journal Iron and Steel Institute," 1901, I, p. 243; II, p. 234.

[2] "Steel," p. 647.

[3] "Journal Iron and Steel Institute," 1902, I, p. 120.

[4] Howe, "Steel," p. 23.

Oxide tint.	Temperature, °C.	Appearance of oil bath.	Corresponding uses of the tempered steel.	C., per cent[1]
White . . .	. . .	. . . . . .	Tungsten steel, *hardest*	Tungsten steel
Pale yellow . .	220	. . . . . .	Lancets	
Straw	230	. . . . . .	Razors, surgical instruments, engravings, taps, dies, cutters	1.5
	232	First smoke		
Golden yellow .	243	. . . . . .	Razors, penknives, hammers, taps, reamers and dies, for wrought and cast iron, copper, brass, etc., cold chisels for cutting steel	1.3
Brown . . .	255	. . . . . .	Cold chisels, shears, scissors, hatchets.	
	260	Strong dark smoke		
Brown, dappled with purple .	265	. . . . . .	Axes, planes, lathe tools for copper.	
	276	Abundant black smoke		
Purple . . .	277	. . . . . .	Table knives, large shears, wood turning and cutting tools, cold chisels for soft cast iron.	
Violet	. . .	. . . . . .	Cold chisels for brass	0.9
Bright blue . .	288	. . . . . .	Swords, coiled springs.	
Full blue . .	293	. . . . . .	Fine saws, augers, etc.	
	304	Can be lighted, but does not continue to burn .	. . . .	0.8
Dark blue . .	316	. . . . . .	Hand and pit saws, cold chisels for wrought iron and copper.	
		Lights spontaneously .	Spiral springs.	
Just visibly red in the dark .	. . .	Burns rapidly	Clock makers purposes, *softest.*	

[1] In steel for which each temper is employed.

The temperatures to which steels are heated in practice in hardening are given by Landis.[2]

Color.	Blue heat.	Red just visible.	Blood red.	Full red.	Dull cherry.	Cherry.	Bright cherry.	Dull salmon.	Salmon.	Good salmon.	Full salmon.	Bright salmon.	Dull yellow.	Yellow.	Scaling yellow.	Yellowish white.
°C.	297	519	611	643	696	718	742	767	819	841	891	940	990	1,089	1,138	1,287

[2] "Iron Age," April 30, 1896, p. 1021.

L. WELDING, CORROSION, ANTI-RUST COATINGS.

SECTION 108. **Welding.**[1] If two pieces of iron, containing less than 1.2 per cent. C but otherwise free from impurities, are heated to a temperature at which they become plastic and pressed one against the other, the surface being kept bright, they will unite and form a weld. The purer the iron and the lower the C-content the easier will it weld. Wrought iron welds better than steel; steel with 1 per cent. C is still readily weldable. S lessens the welding power, but the amounts usually found in wrought iron or low-carbon steel are too small to do any serious harm at the white heat to which they have to be brought to become plastic. The effect of other elements is uncertain. According to the form of the pieces to be welded there are distinguished the *Scarf Weld* (Figure 315). The ends are upset to thicken them and hammered to form a widely stepped scarf. The *V* or *Fork Weld* (Figure 316), suited for heavier pieces. One end is formed to a *V*, the other to the form of a tongue.[2] The *But Weld* (Figure 317). The ends are thickened, jumped together in the fire, and then hammered. Pieces to be welded should be thicker across at the weld than the finished piece is intended to be, to allow for hammering together. The temperature is highest (a white heat) for wrought iron, lower for steels. In order to remove the oxide formed in heating the iron and to produce contact of pure metal, fluxes, such as sand for wrought iron and borax for steel, are added which will slag the iron oxides. The slag coating the clean iron surface will be squeezed out, while the plastic surfaces are forced together by hammering. The hammering is continued until the piece has cooled below a red heat, as it removes much of the coarse crystallization due to heating well above *W* (Figure 312). A welded piece of wrought iron is, as a rule, not so strong as when it has not been welded; a welded piece of steel never is as strong.[3]

In electric welding two processes, those of Thompson and Bernardo, have to be distinguished. In the Thompson process[4] the two pieces to be welded are held in copper clasps and pressed together end to end, while a current of very high amperage and low voltage heats the ends by the resistance that the uneven surfaces offer to it. The ends are made convex, and, softening under the heat, become slightly upset. A little borax is added to the joint to slag any oxide formed. In the Bernardó process an electric arc melts chips of metal of the same composition as that which is to be welded between the ends to be united. It is a soldering process. The pieces to be welded form the negative pole, a carbon pencil the positive pole.

In the Goldschmidt process[5] the parts to be welded are cleaned, butted against one another, and the joint surrounded by "thermite," a paste made up of a mixture of Fe_2O_3, granulated Al, and some adhesive substance. On top of this thermite is placed a primer consisting of BaO_2 and finely divided Al. The primer is ignited by means of a piece of Mg wire. The heat generated by the oxidation of the Mg brings the primer to the reaction temperature, when BaO_2 oxidizes the finely divided Al. The heat set free is sufficient to start the oxidation of the granulated Al by means of the O of the Fe_2O_3. When once started, the process goes on of its own accord; Fe_2O_3 is reduced to Fe, and Al oxidized to Al_2O_3 and fused, the tem-

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[1] Howe, "Steel," p. 250; Campbell, "Manufacture and Properties of Iron and Steel," 1903, pp. 26, 139, 583.

[2] A fork weld, etched to bring out the structure, is published in the "Iron Age," January 28, 1898, p. 13.

[3] Beutner, Manufacture of Welded Pipe: "Iron Trade Review," December 17, 24, 1903; "Iron Age," February 4, 11, 18, 1904.

[4] E. Thompson, Electric Welding Development: "Cassier's Magazine," 1904, XXXVI, p. 225.

[5] "Technology Quarterly," 1902, XV, p. 101.

perature reaching 3,000 to 4,000° C. In order to retard the reaction and to furnish a desired amount of slag, the thermite paste is diluted by substances such as SiO_2, CaO, and MgO. These combine with the Al_2O_3 to a slag which encloses and heats the welding joint, while the reduced Fe sinks to the bottom. The two parts thus brought to the required welding temperature are pressed one against the other and welded under the slag cover.

SECTION 109. **Corrosion.** Literature: Simon, "Ueber Rostbildung," Gewerbefreund, Berlin, 1896; Andés, "Der Eisenrost," Hartleben, Vienna, 1898; Wood, "Rustless Castings, Corrosion, and Electrolysis of Iron," New York, 1904; Howe, "Steel," p. 94; Sexton, "Metallurgy of Iron and Steel," Manchester, 1902, p. 570; Wedding, "Eisenhüttenkunde," Berlin, 1901, p. 362; Parker, Relative Corrosion of Iron and Steel: "Journal Iron and Steel Institute," 1881, I, p. 39; Muck, Corrosion of Boilers: "Stahl und Eisen," 1888, p. 837; Heyn, "Mittheilungen aus den Königlichen Versuchsanstalten," 1900, p. 38; Abegg, Electrochemie des Eisens: "Stahl und Eisen," 1901, p. 736; Rudeloff, Experiments on Rusting of Iron and Steel: "Mittheilungen aus den Königlichen Versuchsanstalten," 1902, Heft 2 and 3; "Stahl und Eisen," 1903, p. 384; Diegel, Behavior of Some Metals in Sea Water: "Verhandlungen des Vereins zur Beförderung des Gewerbefleisses," 1903, LXXXII, B. 157; "Stahl und Eisen," 1904, pp. 567, 629; Dunstan, "Proceedings of the English Chemical Society," 1903, XIX, pp. 150, 157; "Stahl und Eisen," 1904, p. 346; "Berg- und Hüttenmännische Zeitung," 1900, p. 471; Andrews, "Iron," 1889, XXXIII, pp. 74, 340, 451; XXXVII, pp. 71, 252; XXXVIII, pp. 270, 423; Norton, "Iron Age," November 6, 1902.

Iron exposed to the action of moist air oxidizes, forming rust, hydroxide of iron containing more or less ferrous carbonate. For iron to rust, the three bodies, O, H_2O, and CO_2, must be present together. Iron will also rust if the CO_2 is replaced by fatty[1] or other acids or certain salts (alkaline chlorides). No general laws can be given as to resistance of iron against rusting, as this depends upon the composition, the structure, and accidental causes (particles of slag, segregation, magnetization of surface) which may give rise to the formation of local currents and cause a quick rusting. The effect of H_2SO_4 upon different forms of iron was given on page 107. Howe[2] investigated the relative corrosion of wrought iron, soft steel, and nickel steel when exposed to sea water, to river water, and to weather for periods of about one year. He finds that the difference in the rate of corrosion between wrought iron and soft steel is rarely enough to be of great moment, except perhaps in marine boilers, and that the ratio of the corrosion of given soft steel to that of given wrought iron may vary greatly with the condition of exposure. A summary of his results is given in the subjoined table.

Kind of metal.	Sea water.	Fresh water.	Weather.	Average.
Wrought iron	100	100	100	100
Soft steel	114	94	103	103
Nickel steel, 3 per cent. Ni . .	83	80	67	77
Nickel steel, 26 per cent. Ni . .	82	32	30	31

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[1] Mercier, "Annales des Mines," 1879, XV, p. 234.

[2] "Engineering and Mining Journal," August 18, 1900.

SECTION 110. **Anti-rust Coatings.** Literature: Wood, "Rustless Coatings, Corrosion and Electrolysis of Iron," New York, 1904; Howe, "Steel," p. 372; Harbord, "Steel," p. 529. Preservative Coatings: "Proceedings American Society for Testing Materials," June, 1904.

(1) *Tinning.*[1] Pure sheet iron (black sheets) is scoured, dipped in acid (pickled), in water (washed), in melted grease, and passed through molten tin, which coats it. As Sn is electro-negative to iron, it protects it only so long as the film remains whole. As soon as it is scratched or broken the iron will rust much faster than if the tin were absent. Steel sheets, after having been pickled and washed, are sometimes stacked horizontally to a height of 2 feet in cast-iron boxes, covered with sand, annealed,[2] cold-rolled to obtain a smooth surface, then reannealed, pickled, washed, and tinned.

If steel sheets 15 x 20 x $\frac{1}{4}$ inches, with 0.16–0.18 per cent. C and 0.020–0.025 per cent. P, are rolled in packages of 2–4 pieces to the gauge required for making tin plate, they are liable to stick together wherever the scale peels off. This does not happen when the P is at 0.04–0.06 per cent. If in open-hearth work the P sinks below 0.020 per cent., apatite (220 pounds) and sand (60 pounds) are added to the ladle to rephosphorize the steel. This mixture has been replaced in recent years by ferrophosphorus with P, 25; Fe, 70; C, 1.5; Mn, 1; Si, 2.[3] The C of the finished plate is reduced to 0.14–0.16 per cent. to compensate for the effect of the P.

(2) *Galvanizing.*[4] The sheet iron is cleaned with HCl, scoured, passed through water, then through molten zinc, covered with a layer of $ZnCl_2$ and NH_4Cl, and the excess of Zn wiped off with asbestos. Zinc is electro-positive to iron; it therefore protects the iron, even after the coating has been injured. Zinc does not adhere as firmly to iron as does tin. An addition of a small amount of Al,[5] or aluminium-zinc alloy,[6] greatly improves its coating power.

(3) *Nickel Plating.*[7] The cleaned iron is put into an electrolytic bath and nickel deposited on it from a solution of nickel ammonium sulphate. Nickel is electro-negative to iron; the coat is hard and does not readily crack or scale off as long as the iron has been made chemically clean and the coat is not too thick.

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[1] W. T. Flanders, "Galvanizing and Tinning," D. Williams Company, New York, 1900; Hartmann, "Verzinnen und Verzinken," Hartleben, Vienna, 1900; Manufacture of Tin Plate: "Iron Age," December 15, 22, 29, 1892; January 5, 12, 1893; Manufacture of Tin Plate (Cronmeyer): "American Manufacturer," September 14, 1899; Practical Tinning: "Stahl und Eisen," 1889, pp. 553, 724, 946, 1006; Manufacture of Tin Plate (Garrison): "Engineering and Mining Journal," 1891, LI, p. 258; Manufacture of Tin Plate (Hammond): "Journal Iron and Steel Institute," 1897, II, p. 24; Manufacture of Tin Plate: "Iron Age," 1895, LVI, pp. 1048, 1092, 1152, 1219, 1266; Manufacture of Tin Plate (Pasquier): "Revue Universelle des Mines," 1899, XLVI, pp. 1, 221; XLVII, pp. 139, 201; Manufacture of Tin Plate (Tregoning): "English Institute of Mechanical Engineers," 1901, p. 1273; Manufacture of Tin Plate (Hammond): "Journal Iron and Steel Institute," 1897, II, p. 24.

[2] Vertical Annealing Furnace for Sheets: "Iron Age," January 25, 1894.

[3] "Iron Trade Review," December 26, 1901; "Iron Age," May 7, 1903.

[4] Books of Flanders and Hartmann just quoted; J. Davies, "Galvanized Iron: Its Manufacture and Uses," Spon-London, 1899; Richter, "Die Galvanische Verzinnung des Eisens," Quandt & Handel, Leipsic, 1895; Richter, Electro-Galvanizing: "Stahl und Eisen," 1895, p. 906; "Zeitschrift für Electro-chemie," 1895, pp. 78–98; "Berg- und Hüttenmännische Zeitung," 1895, p. 155; Cowper-Cowles, Electro-galvanizing: "Iron Age," 1894, LIII, p. 543; "Cassier's Magazine," 1898, XIII, p. 306; "Fielden's Magazine," 1900, II, p. 603; Wood, Galvanizing: "Iron Age," December 13, 1894; Flanders, Galvanizing: *op. cit.*, February 27, 1896; Reese, Improvements in Galvanizing: *op. cit.*, September 30, 1897; Preu, Practical Galvanizing: "Oesterreichische Zeitschrift für Berg- und Hüttenwesen," 1889, pp. 447, 457, 469; Cowles, Galvanizing: "Electro-chemist and Metallurgist," 1904, III, p. 826.

[5] Richards, "Aluminium," 3d ed., p. 529.

[6] "Journal Franklin Institute," 1901, CLI, p. 445.

[7] W. G. McMillan, "A Treatise on Electro-metallurgy," Griffin, London, 1895; A. Watt and P. Arnold, "Electro-deposition," Lockwood, London, 1902; Langbein-Brannt, "Electro-deposition of Metals," Baird, Philadelphia, 1902.

(4) *Russia Sheet Iron.*[1] The process of giving the iron the bluish Fe_3O_4 coat which protects it against rusting is probably the following: pure sheet iron is wetted, dusted with charcoal, piled, heated, and hammered. This is repeated several (four) times. Water has an oxidizing, C, CO, and H have a reducing effect. The result is a coating of Fe_3O_4.

(5) *The Bower-Barff Process*[2] coats cast iron or wrought iron with Fe_3O_4 by exposing it in a furnace for about forty minutes to an oxidizing atmosphere and then twenty minutes to a reducing flame.

(6) *Tar.*[3] Cast-iron pipes, wrought-iron and steel wire ropes exposed to acid mine water are coated with tar.

(7) *Paints*[4] ground with linseed oil are used as a coat for structural iron. The oil forms on oxidation a rubber-like substance having a great preserving power. Red lead aids the oxidation.

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[1] Percy, "Iron and Steel," p. 730; "Journal Iron and Steel Institute," 1871, II, p. 115; Thurston, "Materials of Engineering," New York, 1898, II, p. 209.

[2] "Transactions American Institute of Mining Engineers," XI, p. 338.

[3] Guettier, "Fondeur en Métaux," Paris, 1890, p. 408; Köhler, "Bergbaukunde," 5th ed., Leipsic, 1900, p. 389.

[4] Thurston, "Materials of Engineering," New York, 1898, II, p. 333; Anti-corrosive Paints: "Engineering," June, 1902; Wood, Rustless Coatings for Iron and Steel Paints: "Iron Age," 1895, XCVI, p. 173; Job, Durability of Paints for Protection of Structural Work: "Journal Franklin Institute," 1904, CLVIII, p. 1; Norton, Corrosion of Steel Frames: "Iron Age," November 6, 1902.

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